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DEPARTMENT OF COMMERCE

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MINING METHODS AT THE EIGHTY-FIVE MINE, CALUMET AND ARIZONA MINING CO., VALEDON, N. MEX.

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MINING METHODS AT THE EIGHTY-FIVE MINES, CALUMET AND ARIZONA MINING CO., VALEDON, N. MEX.[1]

By Ralph B. Youtz[2]

INTRODUCTION

This paper describing mining methods at the Eighty-Five mine of the Calumet and Arizona Mining Co., Valedon, N. Mex., is one of a series being prepared by the United States Bureau of Mines on mining methods, practices, and costs in the various mining districts in the United States.

The Eighty-Five mine is in southwestern New Mexico, 3 miles southwest of Lordsburg, Hidalgo County (see fig. 1). The mine produces normally 7,000 to 8,000 tons per month of silicious copper-gold-silver ore, and employs approximately 200 men underground and 40 on the surface. Mining is done by shrinkage and cut-and-fill methods.

ACKNOWLEDGMENTS

In the preparation of this paper the writer acknowledges the assistance given by A. J. Balmforth, superintendent, J. C. B. Amos, chief clerk, and Dale Underwood, mining engineer, of the Eight-Five mine, and by C. H. Johnson, assistant mining engineer of the Bureau of Mines, without whose cooperation this article could not have been written.

HISTORY OF THE DISTRICT

The Eighty-Five mine is in the Virginia mining district, which comprises a large area 3 to 10 miles southwest of Lordsburg in the Pyramid Mountains. The group of 50 patented and 7 unpatented claims owned by the Calumet and Arizona Mining Co., Eighty-Five mines, includes an area of 799 acres in the north end of the mountains. The mine and the townsite of Valedon are connected with the main line of the Southern Pacific Railroad by a standard-gage spur from Lordsburg.

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2 One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, Eighty-Five Mines, Calumet and Arizona Mining Co.

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The Virginia mining district was first prospected about 1870. The Superior claim, now one of the Eighty-Five group, was located in 1875 by Morris Lesinsky and Isaac A. Cohen. In 1880 the mining camp of Pyramid (Leidendorf) was located, 8 miles southwest of Lordsburg. Rich silver ores were found in the Viola (Venus) and Last Chance mines.

In 1881 the railroad was built through Lordsburg. This stimulated mining in the district to such an extent that the same year a smelter was constructed at Shakespeare 2 miles southwest of Lordsburg. In 1882 a 20-stamp, pan-amalgamation mill was erected at Pyramid; it operated for 10 years.

The Eighty-Five claim was located in 1885 by Samuel Ranson. Several years later it was acquired by Joe Leahy of Lordsburg. In 1906 a Mr. Black of Denver bought the Eighty-Five group of claims from Leahy for $40,000. He in turn sold the ground in 1913 to Warner Brothers, Yates, and Barclay, who organized the "85" Mining Co. This company built the townsite of Valedon and the railroad spur from Lordsburg to the mine. On May 1, 1920, the Calumet and Arizona Mining Co. bought the property from Warner Brothers, Yates, and Barclay.

PRODUCTION

Prior to 1913, when the Warner Brothers acquired the Eighty-Five mine, about 100,000 tons of copper ore had been shipped from the upper levels of the mine. From 1913 to May 1, 1920, about 520,000 tons was shipped averaging 0.14 ounce gold, 3 ounces silver, and 3 per cent copper.

From May 1, 1920, to July 1, 1930 783,138 tons, averaging 0.111 ounce gold, 1.23 ounces silver, and 2.79 per cent copper, was shipped to the Douglas smelter of the Calumet and Arizona Mining Co. Production during 1930 continues at the rate of 7,000 tons per month.

The success of the mines has depended on the need of silicious fluxing ores at nearby smelters. The ore produced from 1913 to date has contained at least 67 per cent silica.

GEOLOGY

All the rocks of the Virginia district are of igneous origin, and consist chiefly of andesite, diorite, and porphyries of similar composition. A coarse-grained, light-colored granodiorite, locally called monzonite, intrudes the andesite and diorite from below. One irregular stock of this monzonite is 2,000 feet in width and 2,600 feet long where cut by the Emerald vein, in which the ore bodies of the Eighty-Five mine occur (see fig. 2). In the western end of the mine several dikes of granodiorite and quartz diorite cut the andesite. On the outskirts of the Eighty-Five property and throughout the district the outcrops of rhyolite dikes are found.

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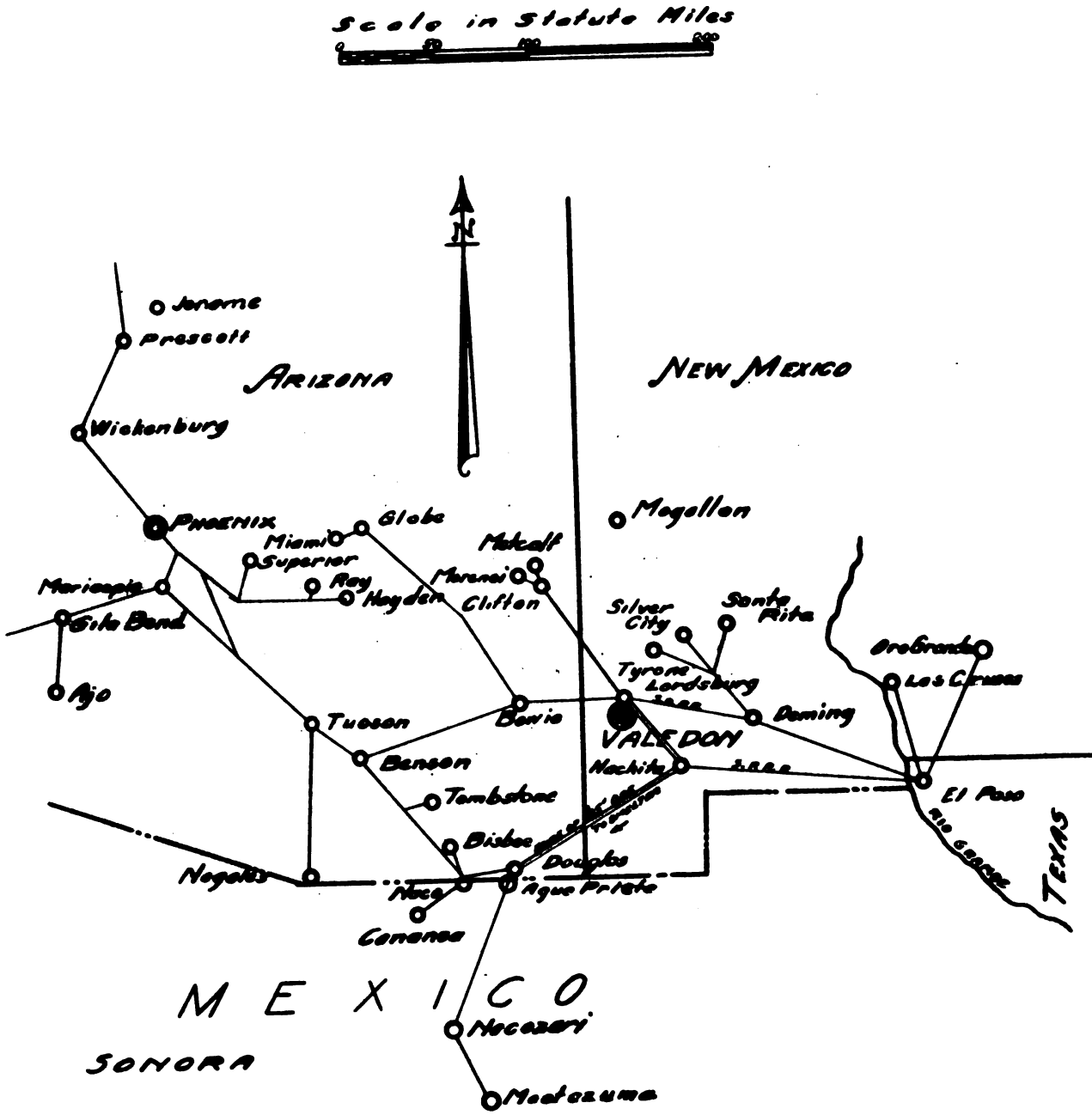

Figure 1.- Map of southern Arizona and New Mexico

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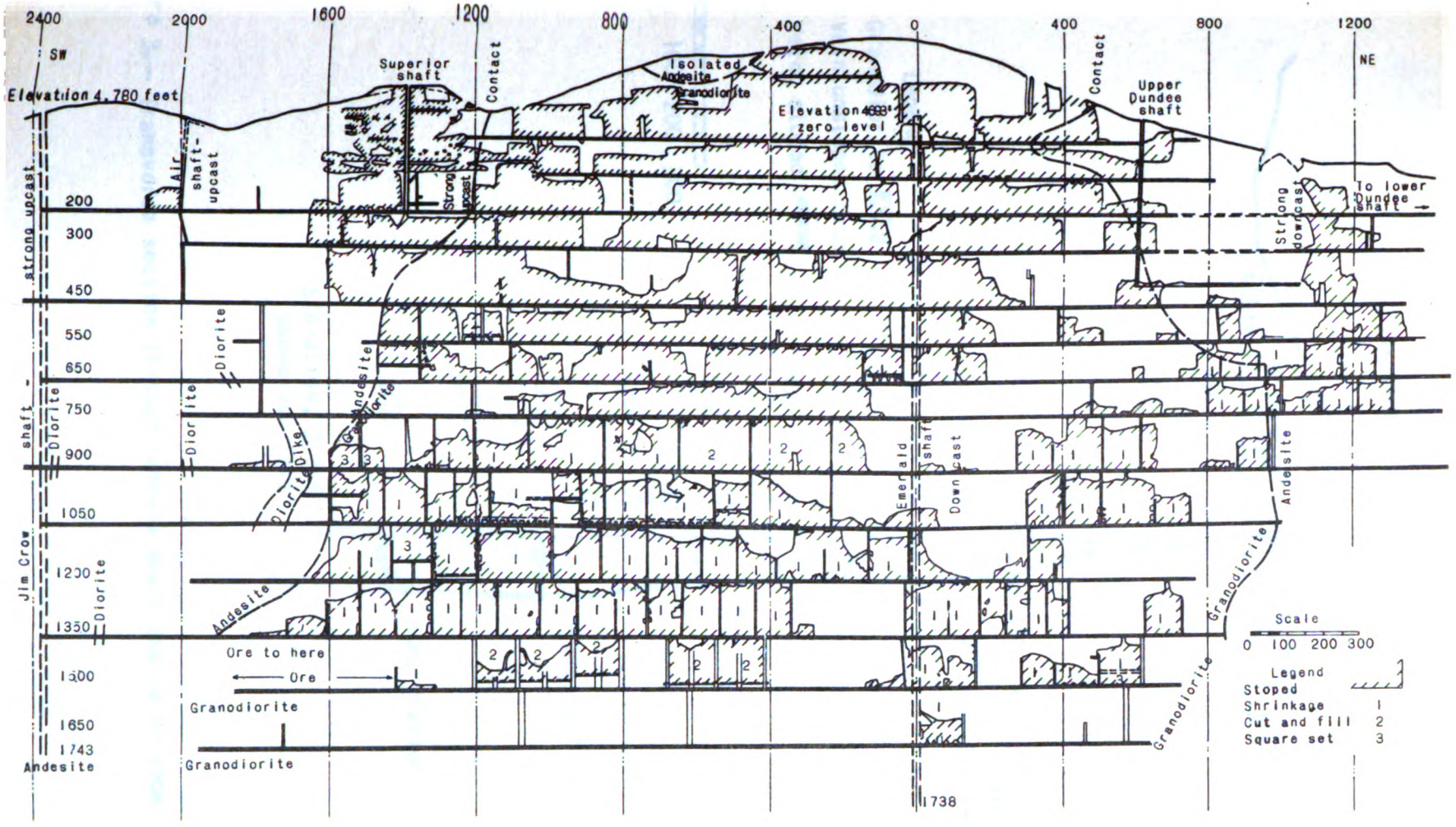

Figure 2.- Longitudinal section of 85 mine

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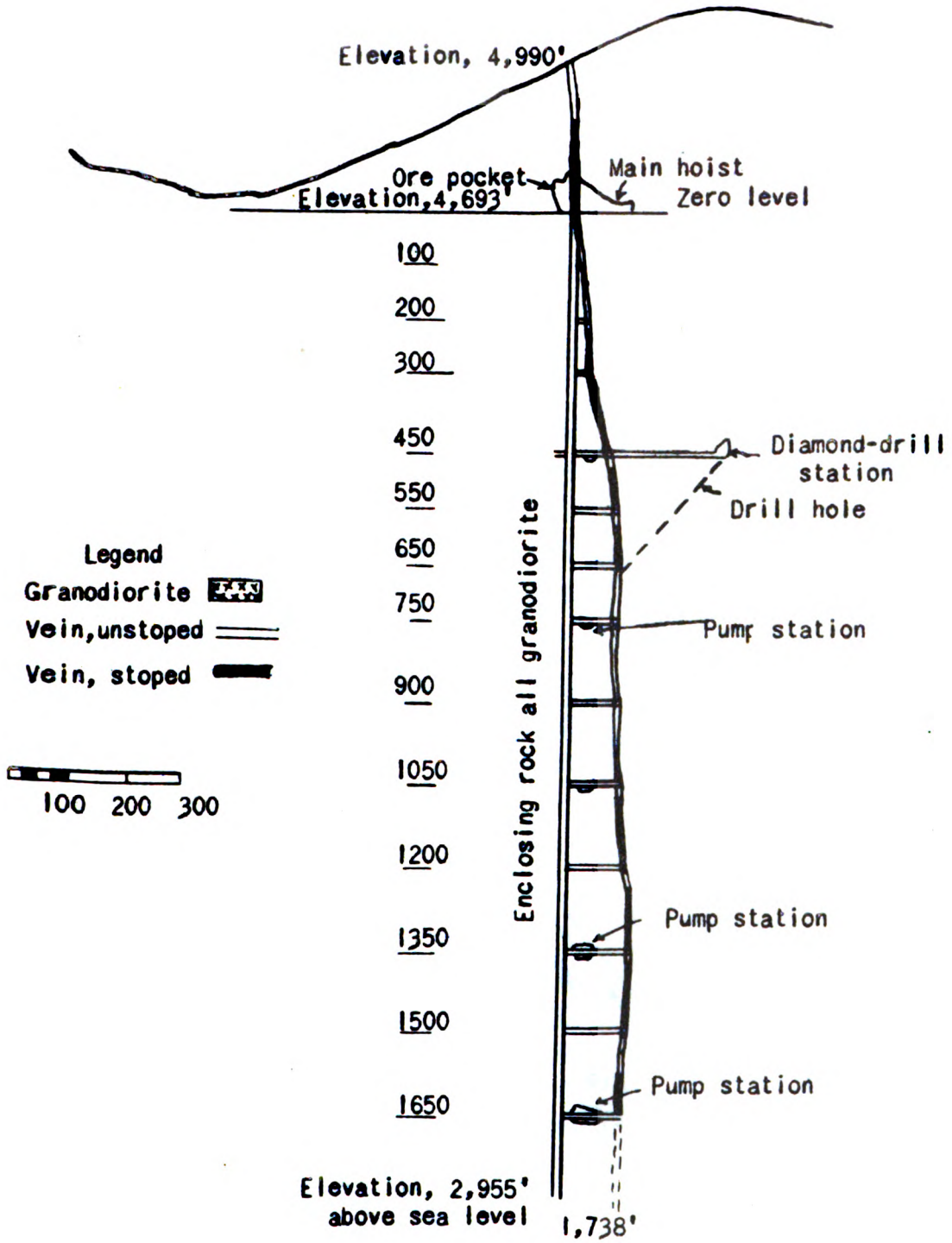

Figure 3.- Transverse section through Emerald shaft and vein, looking north

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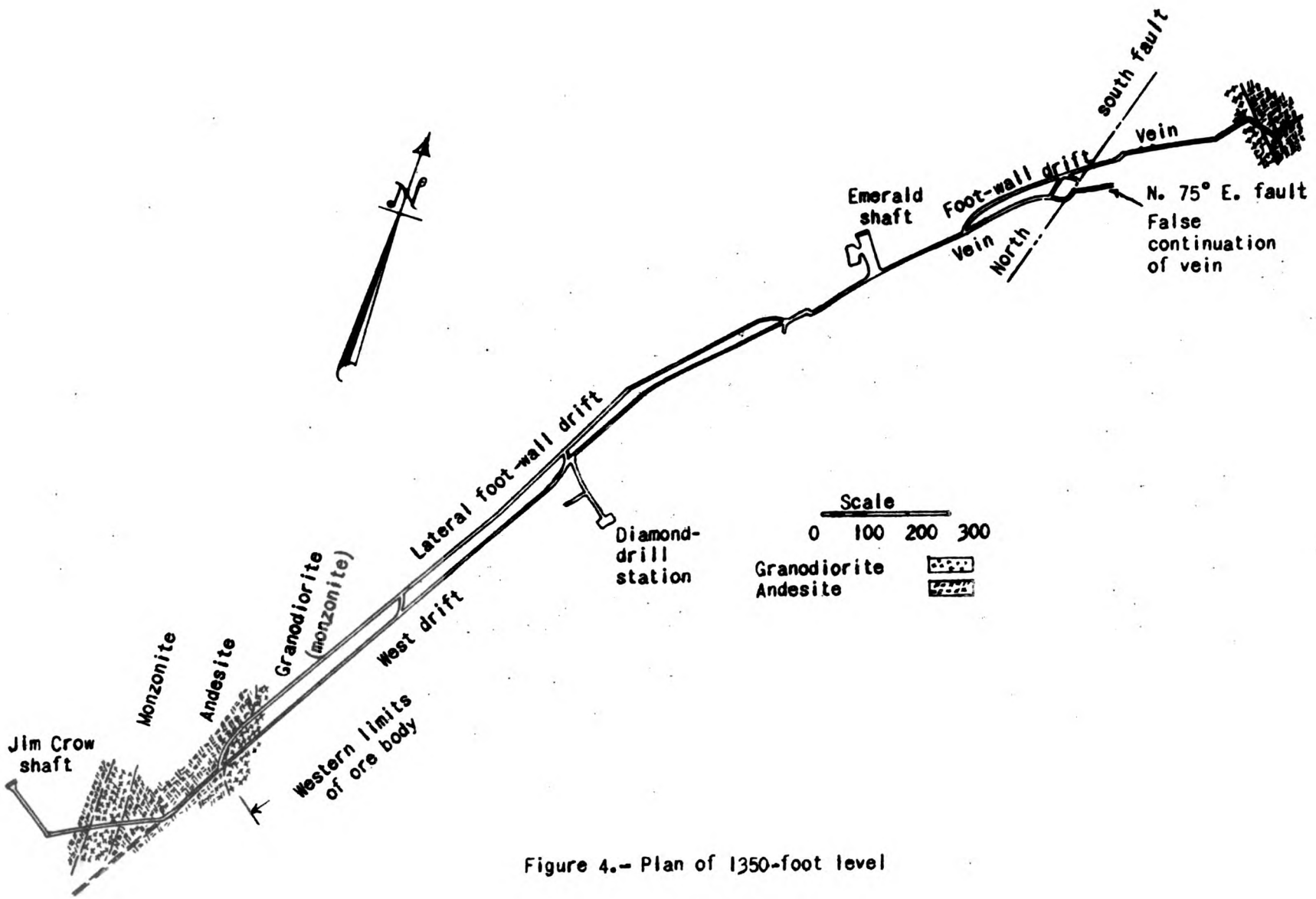

Figure 4.- Plan of 1350-foot level

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A series of fractures of at least three separate systems traverses all three principal rocks. The earliest system and the most important economically has a N. 40 to 45° E. strike. The Emerald, Nevada, Old 85, Carlos, Oakley, Bonney, and Miser's Chest are veins belonging to this group. A later system striking from N. 75° E. to due east, with a steep dip to the south, cuts and frequently offsets the first. The veins of the district are mineralized fault fissures belonging to these two systems. The third system, the most recent, strikes N. 10° W. and faults both older systems.

The northeast system of veins apparently includes the only veins of economic importance, and the Emerald vein is the only member of the group that is mineralized continuously enough to make a large mine. It is a true fissure vein, striking N. 45° E., and dipping an average of 80° to the southeast. The dip tends in places, especially in the east side of the mine, to become vertical or even to reverse steeply to the north (see fig. 3). The vein can be traced on the surface for a distance of 5,000 feet. As shown in Figure 2, ore has been mined from it almost if not quite to the surface for a distance of 2,300 feet along the strike. On the 1500 level ore occurs for a distance of 2,600 feet along the vein. The ore has been followed to a depth of 2,000 feet. The richer ore shoots tend to rake slightly toward the west. The vein in its course through the granodiorite is very regular; it ranges in width from 2 to 10 feet, with an average of 5 feet. At times, especially on the upper levels, the vein attains a width of 30 feet. Where the vein extends into the andesite in both directions along the strike, however, it tends to finger out; the ore bodies found in andesite are very irregular and are "spotty" in value.

The mineralization was probably the final result of magmatic differentiation of which the diorite, andesite, granodiorite, and rhyolite were successive stages. Physically the ore bodies are closely related to the granodiorite stock, as shown by Figures 2 and 4. More than 93 per cent of the ore to date has been mined where the granodiorite formed one or both walls of the vein.

The vein filling consists of massive quartz, country rock replaced by sulphides, and silicified, altered, quartz-seamed rock containing stringers of chalcopyrite and pyrite. Small amounts of calcite, hematite (specular and massive), barite, and rhodocrosite are included in the gangue.

Chalcopyrite and pyrite are the principal primary ore minerals. Galena and sphalerite are found in scattered amounts on all levels. The distribution of values in the primary ores is remarkably uniform; the mineralization and the grade of these ores have varied little in the 2,000 feet of depth developed. The ore limits within the vein are usually commercial, although some barren stretches of vein occur.

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In the oxidized zones azurite, malachite, chalcocite, and chrysocolla have been the principal copper minerals; native copper, cuprite, and covellite were not uncommon. Oxidized ores were found chiefly above the zero level, now the main haulage adit and entrance to the mine. West of the main or Emerald shaft very little oxidized ore was found below the 200 level. East of the shaft, oxidation and leaching are unusually erratic and at times remarkable complete. The porous, iron-stained quartz, containing secondary chalcocite, makes important ore bodies as deep as the 1500 level, often under hundreds of feet of primary ores. Some cuprite and covellite are likewise found as low as the 900 level. In the leached areas the silver content is usually higher than in the unaltered zones. The gold content is very erratic; stopes averaging 0.50 to 0.75 ounce per ton are sometimes mined directly under others averaging 0.08 to 0.10 ounce.

The vein material composed of quartz, silicified country rock, and sulphide is hard and stands without supports over wide spans. On rare occasions the hanging wall slacks or sloughs off to such an extent as to weaken the stope back, which may then break down in large blocks. This appears to take place chiefly in the stopes near the andesite-granodiorite contact in the west side of the mine. The ore is hard to drill, out breaks finely upon blasting, so that secondary blasting is seldom necessary in the stopes.

In the upper part of the mine, both hanging and footwall are firm and will stand unsupported for considerable periods. Below the 1350 level the walls are more mineralized and quite frequently thoroughly kaolinized. In some places this condition of the wall causes cut-and-fill mining to be used instead of shrinkage stoping.

Postmineral movement along the vein, following its offsetting by the east-west faults, produced false continuations of the main vein which were then mineralized by later and weaker solutions. Thus the main vein, as shown for example in Figure 4, at the east end, may appear to be pinching to a stringer, when in reality it has been faulted to the north. Therefore, in drifting and even in stoping, the main ore shoot is often lost unless operations are closely watched.

METHODS OF PROSPECTING AND EXPLORATION

Underground diamond drilling was done by the old 85 Mining Co. and also by the present management up to 1924. Horizontal holes were drilled into both foot and hanging walls to locate parallel veins. On the 450, 750, and 1350 levels, on both the east and west sides of the mine, crosscuts 150 to 250 feet long were driven into the hanging wall at right angles to the vein. From the ends of these crosscuts, diamond-drill holes were drilled toward the vein to intersect it at depths from 150 to 500 feet vertically below the drilling level. From 2 to 4 holes were drilled from each crosscut. Diamond drilling totaled 9,049 feet.

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As diamond drilling and later development work showed the dip of the vein to be uniform and the ore shoots to be fairly persistent, prospecting is now done by sinking the shaft to a new level, crosscutting to the vein and drifting thereon.

Crosscutting is practiced extensively in both the extreme east and west areas to explore for parallel veins and splits of the known vein. Raises are driven where the outlines of ore bodies are unknown. Whenever possible the aim is to make preliminary prospecting serve for ore development and stoping.

METHODS OF SAMPLING AND ESTIMATION OF ORE BODIES

Grab samples are taken from all development faces in mineralized ground under the direction of the shift bosses. They are taken either by the muckers from the pile at the face, or by the "grizzly man" at the shaft station. Stope samples also are taken by the latter from each car as it is dumped into the pocket at the shaft.

In addition to the sampling mentioned, an engineer helper samples all development faces. If the ore encountered is marginal in value it is sampled by cutting channels across the back of the drift or raise with moil and hammer at 5-foot intervals. If the ore is high-grade the channels are spaced at 10-foot intervals. The engineer helper also takes 50-pound grab samples of the muck pile whenever possible. Stope faces are sampled only when the values are marginal or to determine grades for the yearly ore estimate. Mine samples are generally 10 to 15 per cent higher than the smelter samples.

All assay returns are recorded on assay sheets. The returns from all faces are plotted on a longitudinal section assay map.

An ore reserve estimate is computed yearly. As the stopes are generally at 100-foot intervals, the estimate is based on 100-foot blocks. The estimate includes two classes of ore - actual ore and probable ore. When the ore has been cut on two consecutive levels and the probable shape of the ore body has been outlined, the ore between the levels is all classed as actual ore. If the ore has been developed by a drift on one level only, the actual-ore outline is drawn 25 feet below the drift, 35 feet beyond the face, and 35 feet above the drift (or the proved height if raising or stoping has been done).

When a drift on the bottom level, or any drift that extends beyond other workings, is in ore, an additional 25 feet is classed as probable ore, below, beyond and above the drift. Where levels are 250 to 300 feet apart and each level is in ore, a similar estimate of probable ore in addition to actual ore is made. Probable-ore estimates resulting from the application of these rules are often reduced because of the erratic nature of the ore bodies in certain sections of the mine, especially in the east side or where the walls are andesite.

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In calculating the grade of ore reserves, proper deductions are made for differences between mine and smelter assays and for dilution. A factor of 14 cubic feet per ton of ore in place and 20 cubic feet of broken ore per ton is used in calculating tonnages. The former is high enough to allow for sorting and for the occasional horses of waste encountered in stoping.

The amount of broken ore in shrinkage stopes is estimated monthly.

METHODS OF DEVELOPMENT AND MINING

Development

General Plan of Development

The Emerald vein is developed from the zero level or haulage adit. The distance from the portal, where the power plant and other surface equipment are located, to the Emerald shaft is 865 feet.

The Emerald is a three-compartment vertical shaft. It is collared in the vein at the zero level, 300 feet below the outcrop. From there it extends to a depth of 1,732 feet below the adit level, the dip of the vein soon placing it 100 feet or more in the footwall (see fig. 3).

When the Calumet and Arizona Co. acquired the mine, levels had been run at 100, 200, 300, 450, 550, and 750 feet below the zero level. Connections had also been made to the Superior shaft to the west, the Upper and Lower Dundee shafts to the east and the Old 85 shaft to the north. All these shafts, although open for ventilation purposes, are now inaccessible to travel.

From the 750 level, which is about the top of the present active zone, down to the 1650 level, the mine is developed by levels spaced 150 feet apart vertically.

The Jim Crow shaft was recently sunk 2,360 feet west of the Emerald shaft in the footwall of the vein (see fig. 5). The Jim Crow is a three-compartment shaft of the same dimensions as the Emerald shaft. It connects at present with the 200, 450, 650, 900, 1050, and 1350 levels. It has been sunk to the 1650 level, however, and will be connected with the 1500 and 1650 levels in the near future. This shaft was sunk for ventilation, emergency exit, and as a base from which to prospect the western area of the mine.

The general plan of development of any one level is as follows: The Emerald shaft is sunk about 10 feet below the rail elevation of the new level, and a shaft station is excavated equal in width to the length of the shaft and 15 feet in length and height. Shaft bearers are placed in position followed by shaft timbering and several station sets. Then shaft sinking and additional station excavation are resumed. A crosscut is driven to the vein and drifts started in both directions along the vein. Raises are driven to the level above at about 500-foot intervals until the limits of the ore bodies are reached. These raises are used for ventilation and emergency exits.

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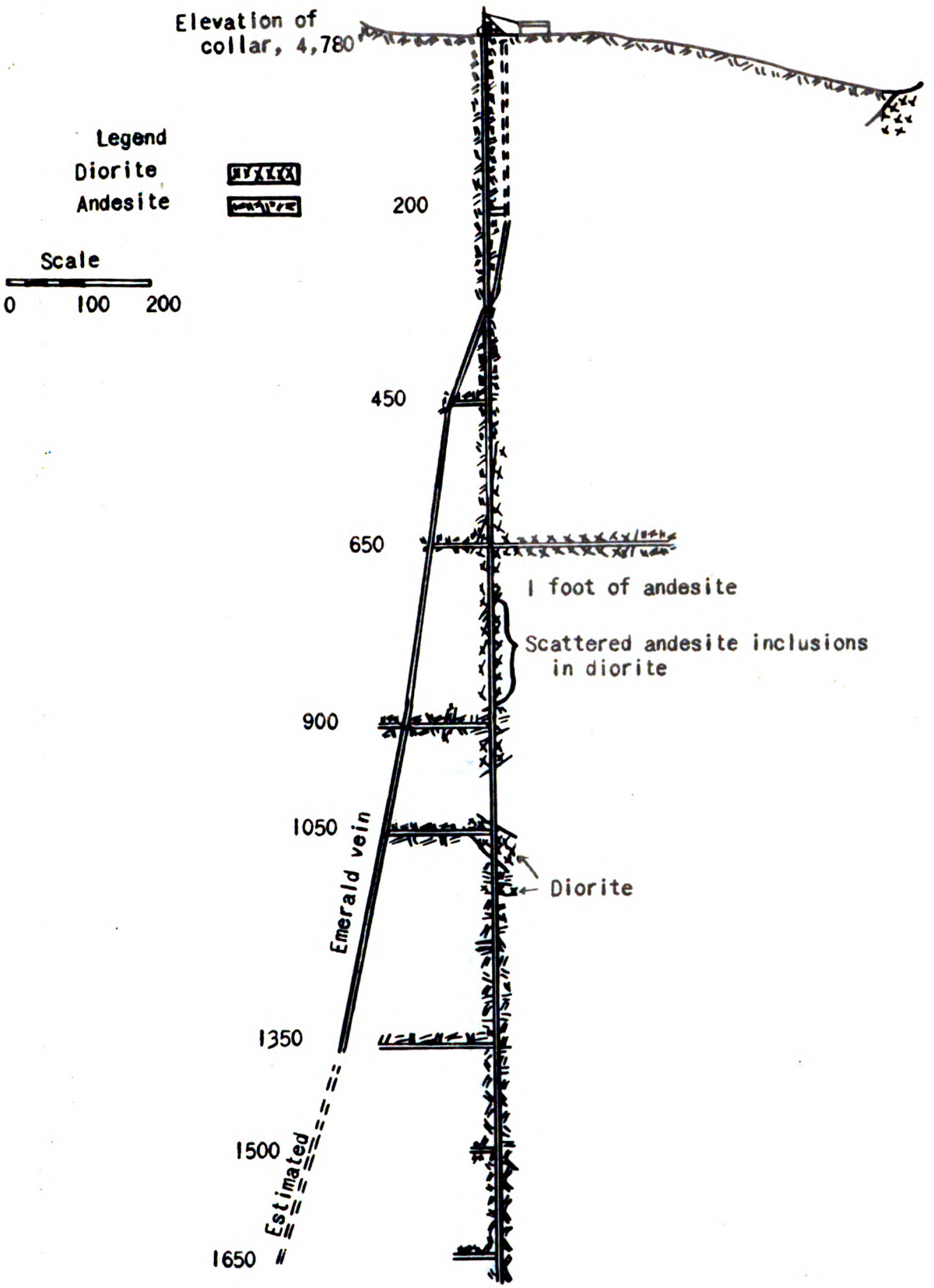

Figure 5.- Section through Jim Crow shaft and Emerald vein

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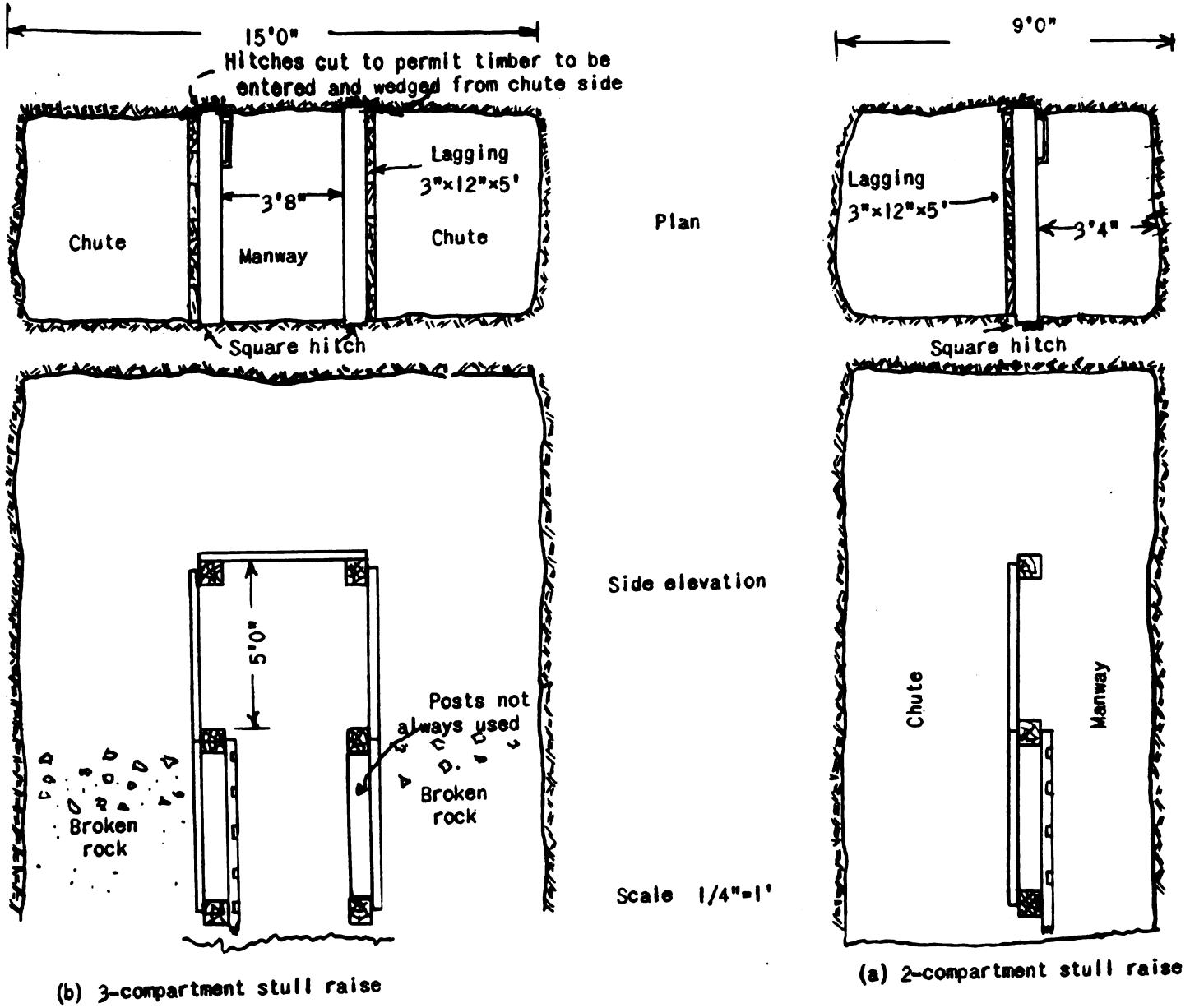

Figure 6.— Stulled raises

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Drilling and Blasting

Drilling and blasting practices are regulated by the mine foreman and shift bosses. A Leyner-type drifter weighing 118 pounds is used in all drifting. Drill columns and crossarms 3½ inches in diameter are used. A crossbar is used only when drilling is done on two or three shifts per day. A 60-pound jackhammer is used for shaft sinking, slabbing, and drilling bowlders in stopes. Hand-rotated stopers are used in stoping and raising operations. Line oilers are standard equipment. The air pressure at the working face is 80 pounds per square inch.

One-inch hollow round steel is used in all Leyners and jackhammers, and one-inch cruciform steel with stopers. Bits are of the cross type, with double tapers of 5 and 14°. The gage of starter bits is 2 inches for both types of steel with a reduction of one-eighth inch for each succeeding steel. Starters are 24 inches long; the change in length is 12 inches.

Forty per cent strength gelatin dynamite, No. 8 caps and waterproof fuse are used in all blasting in drifts, raises, and stopes. Electric blasting is practiced in shaft-sinking operations.

Drifts

Drifts are generally 5 by 7 feet in cross section, although if the vein is 6 or 6½ feet wide the drift is run that width. If the vein width is greater than 6½ feet, the drift is run on the footwall. Often the higher-grade ore is along the footwall and the stope chutes are spaced along the footwall. Prospect crosscuts are 4 by 6 feet in cross section. A standard V-cut round is used. The number of holes in drift rounds ranges from 13 to 22, depending on the hardness of the vein material. Timbering is rarely necessary as the ground is very firm. All mucking is done by hand.

Raises

Raises are driven on the vein between the levels for prospecting, developing, stoping, and ventilation. Raises are of three classes, depending on the use to which the raise to is be put and the nature of the ground.

1. Stull raises with two compartments, a manway and a lined chute (fig.6). The cross section of this type of raise is 5 by 8 feet. However, if the vein is from 5 to 7 feet wide the raise is driven the width of the vein. If the vein is more than 7 feet wide the raise is driven along the hanging wall in order to hitch the stulls more securely into the wall. The hanging wall generally carries some gouge, and ore left on the hanging wall will slough off more readily than ore left on the footwall. The manway can later be used in stoping operations by installing a timber slide and an air line. This type of raise is used for prospecting, ventilation, and in developing shrinkage stopes.

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2. Stull, three-compartment raises, with a manway and two lined chutes (see fig. 6 b). This type of raise is used for ore and waste transfer and for developing cut-and-fill stopes.

3. Cribbed, two-compartment raises. The cross-section of the chute compartment is 4 feet 2 inches square and that of the manway 4 feet 2 inches by 2 feet 6 inches. The cribbing is generally of 2-inch timber. This type of raise is used in bad ground.

Shaft Sinking

The two main shafts, the Emerald and the Jim Crow, have three compartments, two for the skips and cage and a manway and pipe compartment. All compartments are identical in cross section, 4 feet by 4½ feet inside the timbers (see figs. 7 and 8).

As sinking methods are the same at both shafts, the operations just completed (September 8, 1930) at the Jim Crow shaft will be described.

The Jim Crow shaft with a rock section of 7¾ by 16⅔ was sunk from the surface (elevation 4,780 feet or 87 feet above the collar of the Emerald) to a depth of 1,742 feet, 26 feet below the 1650 level through andesite and diorite.

The sinking crew on each of three consecutive 8-hour shifts consisted of one jigger boss and three miners (all at the bottom of the shaft), one topman and one hoisting engineer. One shift boss had charge of the entire crew, which consisted of a total of 18 men per 24 hours. Mexican labor was used in the shaft. The shaft sinking was done on contract.

The time cycle of operations at the 1650 level for a 5-foot round which broke about 70 buckets of rock was as follows:

1 hour setting up equipment,
6 hours drilling,
1 hour removing equipment and loading,
1 hour waiting for smoke to clear out,
12 hours mucking and picking bottom,
5 hours timbering.

When drilling operations were started a manifold with six air connections, four for jackhammers and two for blow pipes, was lowered through No. 1 or the manway compartment. When the drilling was finished the manifold was hoisted above the bulkhead in the same compartment. Steel and supplies were handled in a bucket suspended under a crosshead in No. 2 compartment.

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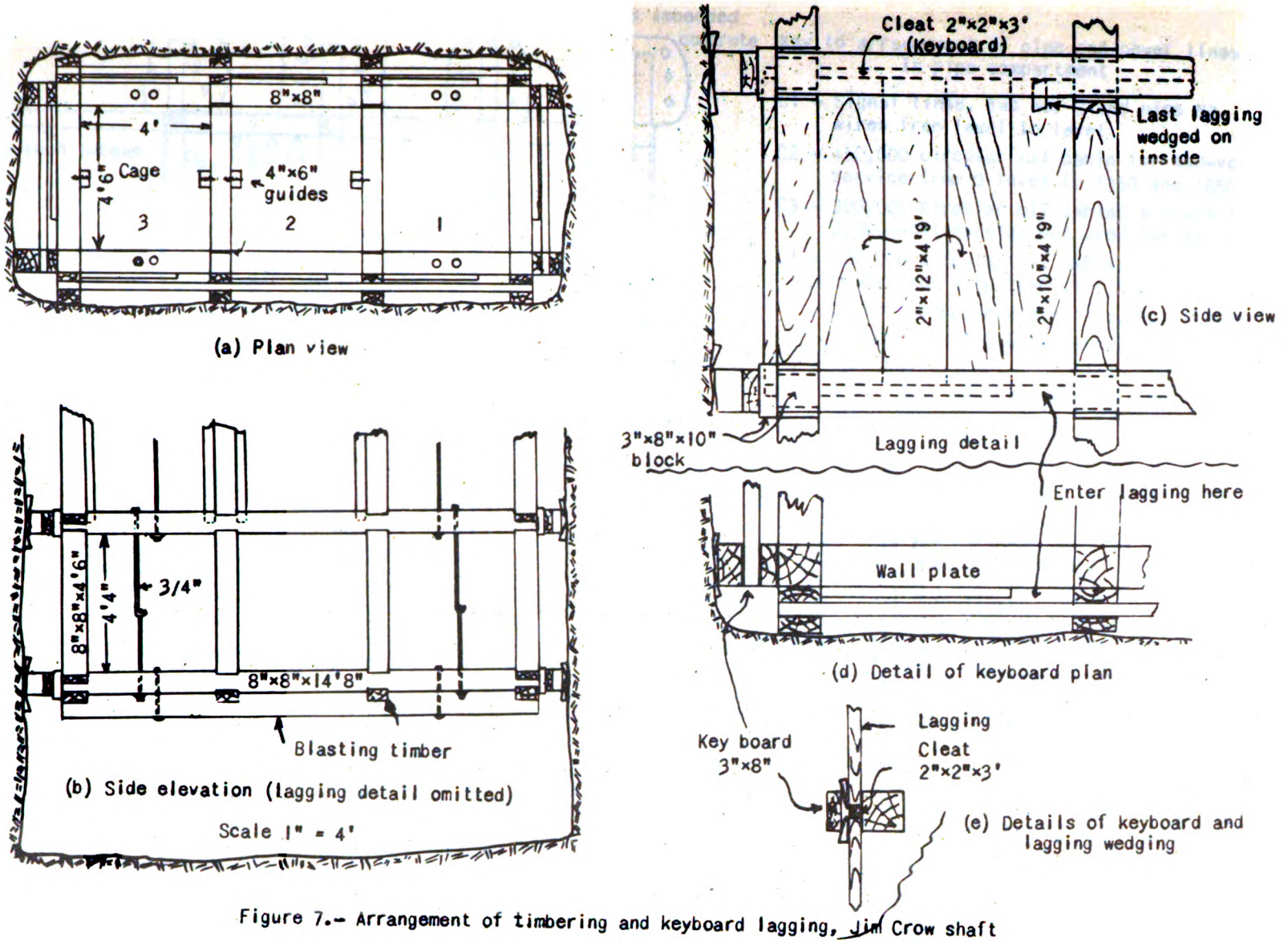

Figure 7.— Arrangement of timbering and keyboard lagging, Jim Crow shaft

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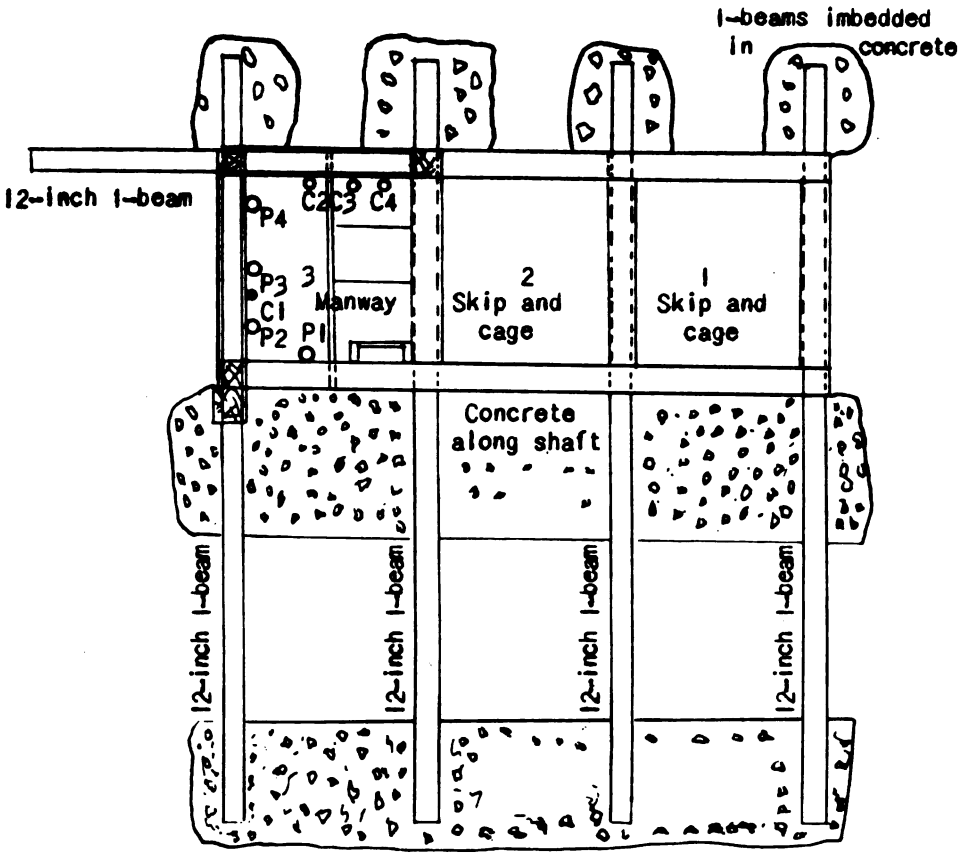


Key to arrangement of pipe and power lines in pipe compartment

C1 – Signal lines, two No. 8 and nine No. 10 wires from level to level

C2 – 450,000 circular mil cable for 440-volt service from 0 level to 1350 and 1650 level

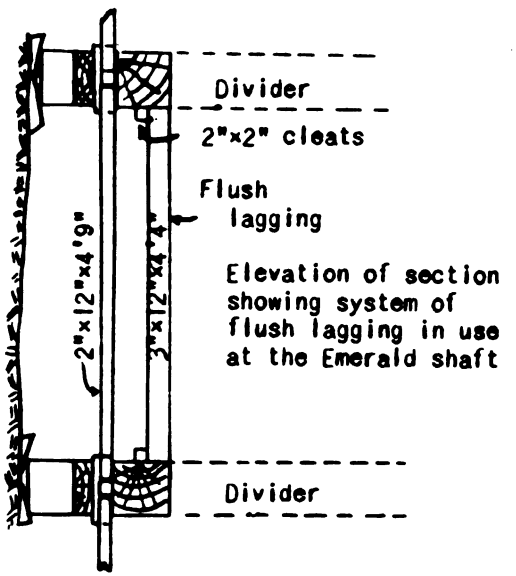
C3 – 300,000 circular mil cable; a spare for 2,300-volt service but used for 440-volt

C4 – 300,000 circular mil cable; 2,300-volt power from 0 to 1650 levels

P1 – 6-inch compressed-air line, to all levels

P2 – 4-inch drinking-water line, from 0 to 750 levels

P3 and P4 – 6-inch mine-water columns, 0 to 1650 levels



Elevation of section showing system of flush lagging in use at the Emerald shaft

Figure 8.– Plan of collar of Emerald shaft

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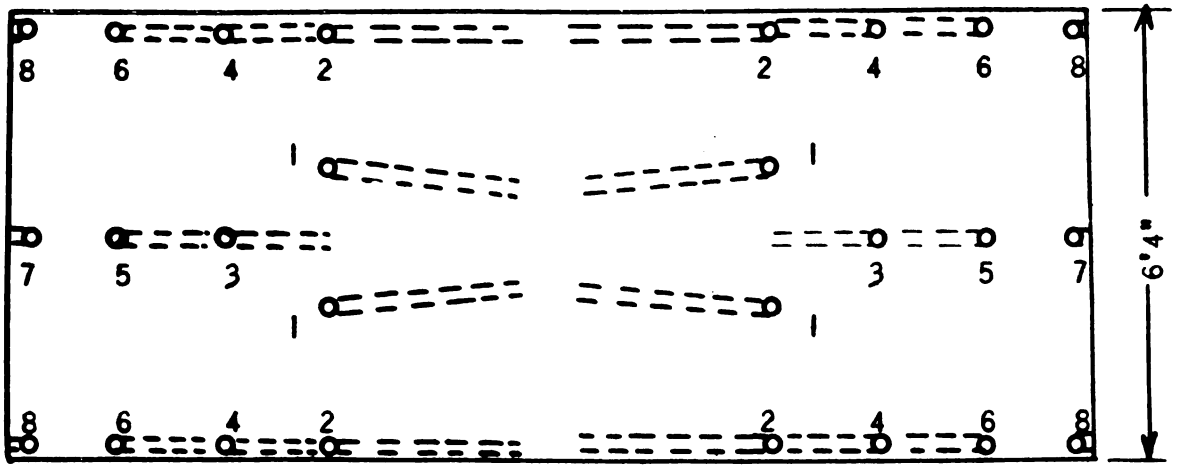
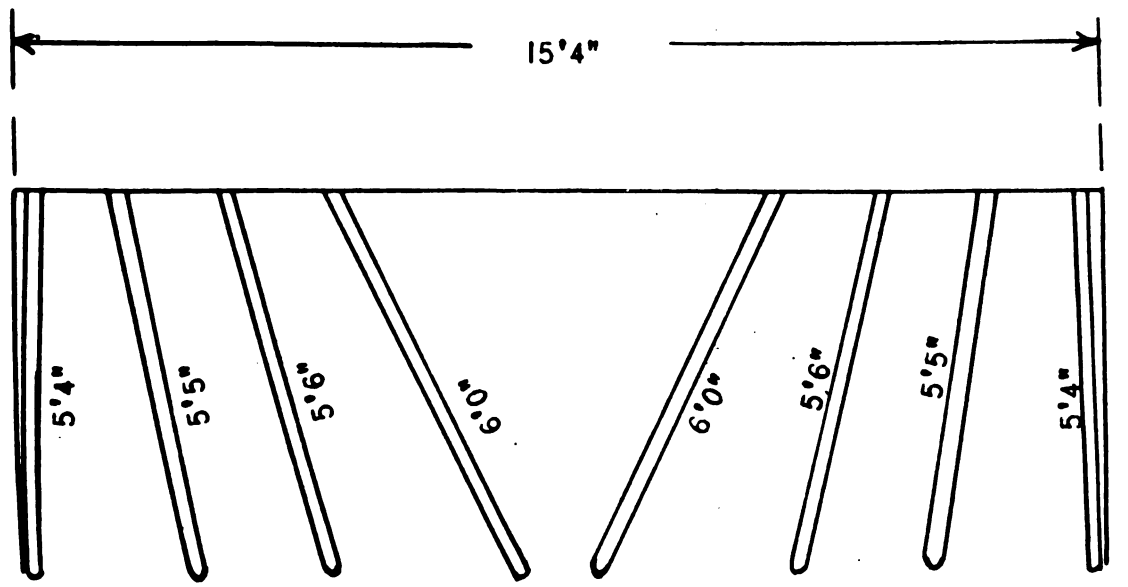


Plan



Side view

Figure 9.- Standard shaft round. Numbers indicate order of firing and electric delay primers used

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A standard V-cut shaft round, consisting of 26 or 36 holes, was drilled with 58-pound jackhammers (fig. 9). The andesite is good drilling ground except when silicified near the contacts. The diorite although very hard is fair drilling but tough breaking ground. Very little heavy or loose ground was encountered. The cut holes were loaded with 5 to 7 1-1/8 by 8 inch cartridges of 60 per cent strength gelatin dynamite and the other holes with 4 to 6 sticks of 40 per cent gelatin. No. 8 electric detonators in 1 to 10 delays connected to a 110-volt light circuit were used for firing the rounds.

The broken rock was shoveled into two 15½ cubic foot buckets which were hoisted in the No. 2 compartment. One bucket was being loaded while the second bucket was being dumped. Above the 900 level all spoil was hoisted to the surface; below this level it was hoisted to a pocket above the 900 shaft station. The broken rock at the 900 level was used for filling stopes. At the 900 and at the surface the buckets were dumped by means of a slide door and tail chain, operated by the topman. About 65 buckets were hoisted per shift. From below the 1650 level where the hoisting speed of the buckets was 450 feet per minute, the hoisting of the rock was the determining factor in the speed of the mucking operations.

Timbering precedes mucking, so that the men can work on top of the muck pile. The timbers are lowered under the bottom of a cage or crosshead in No. 3 compartment.

Shaft timbers are 8 by 8 inch, rough Oregon fir. Shaft sets are on 5-foot centers. Shaft timbers with keyboards and lagging cleats attached are lowered through No. 3 cage compartment. A blasting set (fig. 7 b) is used to protect the shaft timbers during blasting. This blasting set is worn out or broken in about six blasting operations. "Keyboard" blocking (figs. 7 c, d, and e) is always used. The keyboard is of 3 by 8 inch timber. When caving ground is encountered 8 by 8 inch timbers are used for additional "bridge" blocking. Some advantages of keyboard blocking over stull blocking are:

1. On blasting, keyboard blocking is rarely knocked out or loosened.
2. A set can be blocked faster and more securely.
3. When a formerly wet shaft becomes dry, keyboard blocking does not loosen and fall out.
4. The lagging is more secure.

The three compartments are "outside" lagged with 2 by 12 inch by 4 foot 10 inch planks. Lagging is placed behind the set as shown in Figure 7, pushed over on the cleat, and wedged at the top and bottom by wedges placed between the lagging and the keyboard. The last lagging placed is wedged on the inside instead of the outside. Thus all lagging is so secured that it can not possibly become loosened and fall into the shaft compartments.

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In the Emerald shaft all compartments are similarly outside-lagged. But, in addition, the two hoisting compartments are also inside-lagged as shown by Figure 8. Prior to the use of inside lagging, the timbers of the shaft sets were badly worn and the lagging was knocked out by falling rock during hoisting operations.

Bearers are placed at intervals of 150 feet.

As the rock temperature below the 1350 level is rather high, an electric blower on the 1050 station of the Jim Crow forces air through a 12-inch metal pipe to the bottom of the shaft. The temperature at the bottom is still 87° F., and the relative humidity is 100 per cent.

When sinking in the Emerald shaft is resumed a small electric sinking hoist will be installed on the lowest working level, the 1650. As No. 3 compartment (see fig. 8) is full of pipes and cables leading to the pumps, hoisting will be done through No. 1 compartment. A bulkhead will be put in No. 1 compartment above the top of the station and an idler sheave hung so that the cable will not hit the station timber. Muck will be hoisted in buckets and dumped by means of a slide and tail chain into the 1650 pocket, from which it will be hoisted to the surface by means of the skip in No. 2 compartment. If the muck is needed for waste fill in the mine, it will be dumped from the bucket into cars at the 1650 level and hoisted in the cages to other levels of the mine.

The following table shows detailed cost figures for sinking the Jim Crow shaft.

Cost of sinking the Jim Crow shaft

Footage to date: 1,720 feet

	Amount	Cost per foot
Labor and bonus	$59,422.13	34.55
Shift boss = 193.47 per month		
Jiggers = 5.00 per shift		
Shaftmen = 4.40 " "		
Top landers = 3.30 " "		
Hoistmen = 5.50 " "		
Air drills	6,068.22	3.53
Explosives	4,883.22	2.84
Framed timber	14,709.14	8.55
Miscellaneous supplies	6,506.17	3.78
Truck expense 1/	1,589.51	.92
Power expense	4,214.56	2.45
	$97,392.95	56.62

1/ All supplies, drinking water, etc., handled by truck.

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Mining Methods

Early Mining Methods

Prior to 1913, when the Warner Brothers acquired the Eighty-Five mines, some square-setting was done in the upper oxidized zone, but the majority of the stoping was done by shrinkage and cut-and-fill. Some of the shrinkage stopes can still be seen from the surface around the old Superior shaft. A few stulls have kept the walls from caving, even though the stopes were mined 30 years ago.

From 1913 to 1920 most of the ore was stoped by an overhand cut-and-back fill method. In this method a portion of the loose hanging wall was first shot down and the waste spread and smoothed horizontally. Flooring was then laid on top of the waste and the ore shot down upon it and mucked into cribbed chutes carried through the waste filling at 25-foot intervals. Shrinkage stopes were tried very rarely. The cut-and-fill method described above required leaving 5 to 15 foot pillars under the levels. At that time it was thought the pillars would be recovered, but caving of the levels occurred and very few of the pillars under the levels were ever recovered.

When the Calumet and Arizona acquired the mine in 1920, Ira B. Joralemon who was the assistant general manager of the Bisbee mines advised the use of the inclined cut-and-fill method; waste for filling was to be obtained from old filled stopes above. This method produces a cleaner ore and eliminates much handling of ore and waste filling in the stopes.

From 1920 to 1930, mining has been done by the following methods:

Shrinkage	72 per cent of the total
Cut-and-fill ..	28 do.
Square-set	Only four square-set stopes in 11 years of mining.

Of the stopes mined by the cut-and-fill method, 64 per cent were in the period 1920 to 1923. Shrinkage stoping gradually superseded the cut-and-fill method.

Present Mining Method

At present the two mining methods at the Eighty-Five mines are shrinkage and cut-and-fill.

The determining factors in the choice of mining methods at this mine are:

1. Character of the walls of the vein.
2. Quantity and mode of occurrence of country rock in the vein.
3. Grade of the section to be mined.
4. Requirement of a fixed monthly tonnage of a certain copper and silica content.

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Shrinkage stoping is always employed where the walls are firm, and if the vein is more than 3 feet wide. Shrinkage stoping is also always used on marginal ore of high silica content as lower costs are secured by the shrinkage method.

At present the inclined cut-and-fill method of mining is employed in some instances for the following reasons:

1. The walls may be weak owing either to:
   (a) The N. 65° E. to due east system of fractures closely paralleling either wall and forming planes of weakness or
   (b) kaolinization of the walls.
2. The vein is wide, but (a) unmineralized country rock occurs as a wide band within the enclosing walls or (b) splits occur in the vein.
3. High-grade ore may occur in a narrow (2 or 3 feet) vein.
4. To obtain a cleaner product.

At present all the ore is hand-sorted as it passes over a picking belt to the railroad cars. Although one shift of sorters can handle the present 7,000 tons of ore per month, this would be impossible if the 1929 schedule of 10,000 tons per month were in effect. However, in case of necessity the ore coming from the 1500 level cut-and-fill stopes could be run directly into the railroad cars without sorting.

Shrinkage Stoping

The method of stope development described in the following paragraphs is that used on the 1350 level, where all the stopes are shrinkage stopes, four of the 20 being active at present.

Ore was encountered in the West drift, on the 1350 level at a point 320 feet west of the Emerald shaft (see figs. 2 and 4). As the ore body had already been outlined on the 1200 level, a two-compartment stull raise (No. 320 raise west) was immediately started. Advance was quickly resumed in the West drift and by the time the drift had advanced 500 feet through high-grade ore, 320 raise west had connected to the 1200 level. For ventilation and emergency exit another raise was started near the face of the drift. This raise, 725 raise west, was so spaced as to be used in the later stope development.

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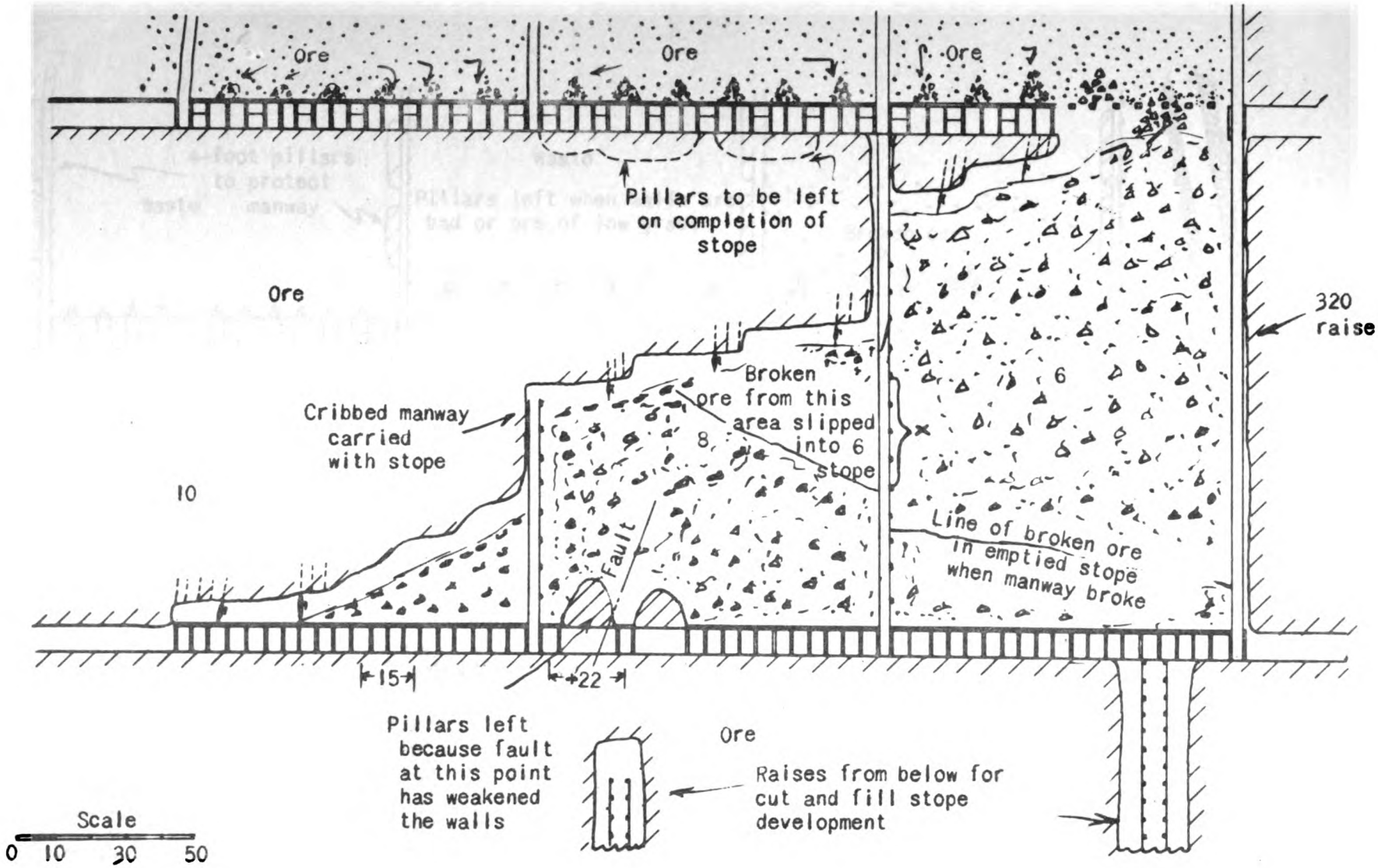

Figure 10.- Shrinkage stope

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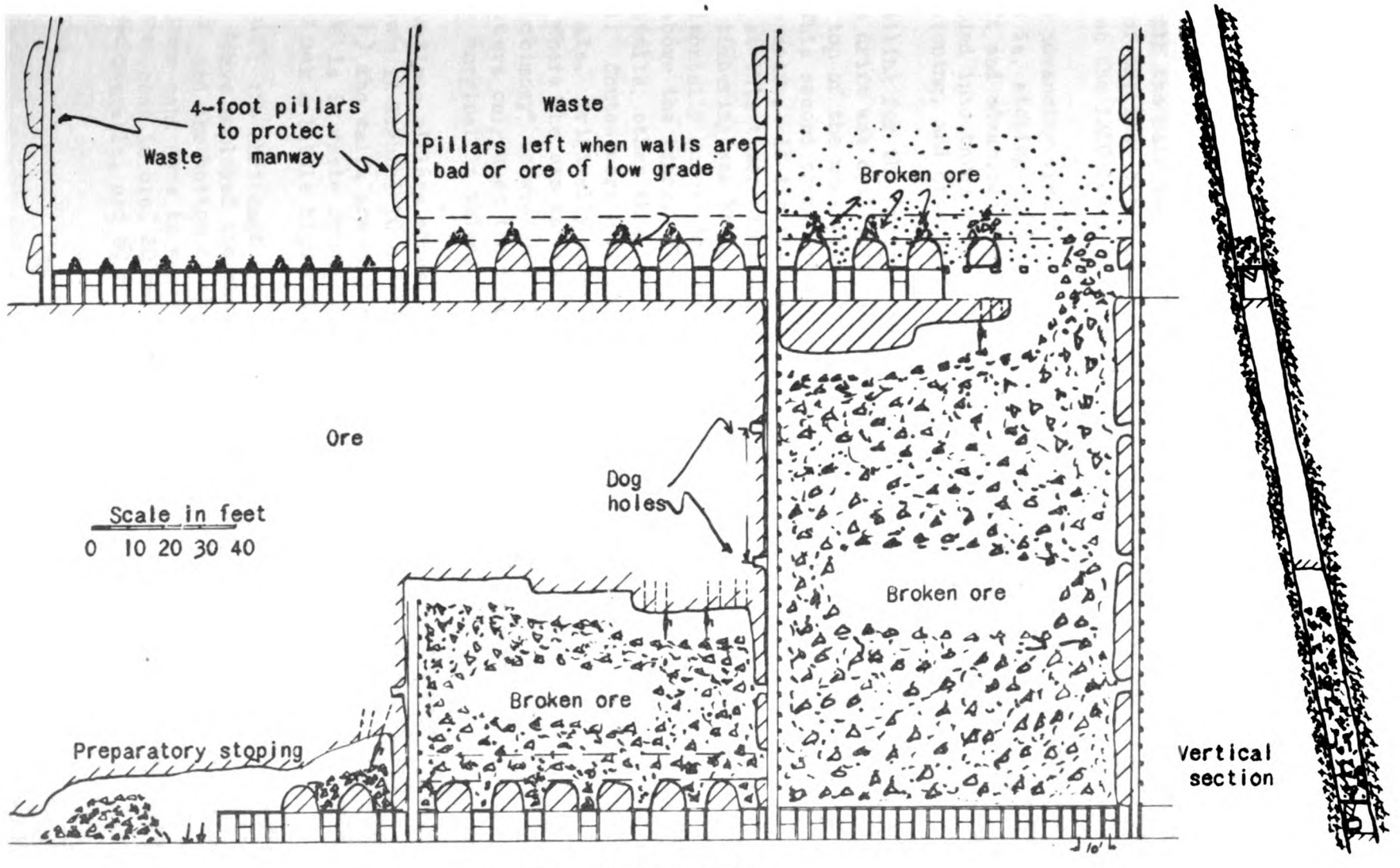

Figure 11.- Shrinkage stope

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As the ore on the 1200 and 1350 levels is high-grade, it was undesirable to leave level pillars under the drifts, therefore the tramming drift was advanced in the footwall. This drift started at the beginning of the ore body and advanced in the footwall parallel to the vein and 25 feet from it. It was connected with the main West drift at intervals and finally at a point beyond the limits of the last ore body. It is used for tramming and travel purposes. This drift on the 1350 level is 1,700 feet long.

The advancing type of shrinkage method is in use at the Eighty-Five mines; that is, stoping operations are started at the end of the ore body nearer the shaft and advanced toward the farther limits of the ore body. Stoping may be divided into three separate operations: Stope preparation or cutting-out, back-stoping, and drawing.

Drilling for the cutting-out rounds was started at the 320 raise. The back of the drift was drilled for some distance and blasted down on the tracks. Working on top of the muck pile the miners drilled a second line of holes and blasted. This second round broke the back to an elevation of 15 feet above the rail of the drift. If the ore was wider than the drift, it was slabbed off to full width at this time. This cutting-out operation now advanced along the drift, and timbering was begun. Starting at 320 raise, 8 by 8 inch stulls were placed horizontally across the drift at 5-foot intervals, and at an elevation of 8 feet above the track. For vein widths of 4 to 8 feet no posts were used under the stulls, other than the chute posts. For greater widths a second post may be used. Chutes were next built on the footwall side of the drift at 10-foot intervals. Originally chutes were placed on 15-foot centers, but it was found that where the ore was wet or selvage occurred within the vein the ore tended to "chimney" between the chutes. Accordingly chutes are now spaced on 15-foot centers only when the material blasted down in preparatory stoping is found to be marginal in value.

Sometimes pillars above the drift are employed in shrinkage stope mining, as shown in Figures 10 and 11. This may be done for either of two reasons: (1) The walls are weak and the vein is unusually wide. (2) The back of the drift is in waste or marginal material but development has shown ore to be in the block a little higher up.

Of all the shrinkage stopes mined during Calumet and Arizona ownership 15 per cent have employed the chute-raise and pillar method, between the back of the drift and the bottom of the stope. Of the pillars thus left above the drifts, 60 per cent were in waste or marginal material and 40 per cent in ore. Of the 40 per cent in ore, 20 per cent have already been mined, 50 per cent are still recoverable, and 30 per cent are inaccessible.

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Back-stoping (advancing the back upward) is next started. Drilling is done with hand-rotated stopers by a crew of 1 to 4 miners. Blasting must be light until the drift timbers are well covered with broken ore. The back of the stope is carried slightly inclined toward the raise or manway that is holed to the level above (320 raise, fig. 10). The angle of the back ranges from 10 to 15° from the horizontal. This assists ventilation and somewhat aids the handling of drills and steel, which are usually brought in from above and taken out to the level below. When the stope can be entered from above at both ends, the back is carried horizontally.

At 100-foot intervals a cribbed manway 2 feet 6 inches by 2 feet 6 inches in the clear is carried up with the stope. This manway is protected on the stope side by stulls set in hitches in the walls at 5-foot intervals (see fig. 10). In narrow veins stulls can be spaced 8 or 10 feet apart. Air lines enter the stope through the manways at both ends of the stope.

As the stope approaches the level above, it may be necessary to leave a 5 to 15 foot pillar below that level. In this case a connection is made in the middle of the stope to the level above, to facilitate the handling of development waste later (fig. 10, stope 8).

If, as is true on the 1200 and 1350 levels, a footwall drift has been driven, no necessity for leaving pillars exists, and the vein is mined out entirely to the level (see fig. 10, stope 6). As the stope above the 1200 level was in high-grade ore, the timbering between the chutes of the worked-out stope is shot out at this time, and the pile of broken ore remaining between the chutes is dropped into the now completed stope below. If pillars of ore were left between the chutes of the stopes above they are now blasted down, as shown in Figure 11.

When the shrinkage stopes are drawn the hanging wall, no longer having any support, begins to slough off. Consequently, by the time the stope is nearly drawn empty, considerable waste has already fallen on top of the broken ore. Therefore, care must be taken that the stopes are drawn down level.

As stated before, the ore breaks rather fine, and if it becomes wet or mixed with the soft material from the walls, it will "chimney" between the chutes. These chimneys of broken ore stand almost vertically to heights of 10 to 30 feet above the bottom of the stope. If the ore is high-grade and no floor pillar is to be left under the level, as was the case on the 1200 and 1350 levels, this broken, chimneyed ore is partly recovered by the following method.

When the shrinkage stope has reached the level and one end is completed, the lagging between the chutes under the worked-out stope above is shot out and the chimneys of broken ore allowed to drop into the completed portion of the stope. Generally the 8 by 8 inch stulls above the drift are not broken, as the blasting is very light. Successively, as the pillar under the level is shot out the lagging between the chutes overhead is also shot, until all the

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ore from the shrinkage stope above is dropped into the lower stope. There is some dilution by waste, but this would occur anyway, for when the stope is drawn the level above will cave whether or not an attempt is made to recover the broken ore. Thus nothing is lost and it is highly probable that some good ore will be recovered. If the ore above was marginal no attempt would be made to recover it.

In the cut-and-fill method of mining, when ore from upper stopes is recovered as above described, a close check on grade and tonnage can be made. It has been found that the first runs of broken ore from above will, even after dilution, average above 2.5 and often over 3 per cent.

The costs of a shrinkage stope worked by the method just described are shown in Table 1.

Table 1.- Shrinkage stope costs, 1350 level, year 1925

(Costs per ton, direct charges only)

	Labor				Other	Air	Total cost
	Bonus	Inc. bonus	Explosives	Timber	supplies	drills	per ton
8 stope	$0.04	$ 0.59	$ 0.23	$ 0.14	$ 0.02	$ 0.39	$ 1.37
6 stope	.02	.75	.27	.20	.02	.46	1.70

Note: The average width of 6 stope was 4.0 feet; of 8 stope, 8.5 feet. Lower costs were obtained in the wider stope.

It is best in shrinkage stoping to have three or four adjacent stopes in successive stages of operation. When back-stoping is finished in one stope, the broken ore should not be drawn out until the next stope in line is completed. If the stope is drawn too soon the weight of the broken ore in the operating stope is liable to push the manway over into the partly drawn stope. An example of this is shown in Figure 10.

During 1928 and 1929 development work on the 1350 west drift was delayed. Repair work in the operating shaft during these years also reduced total mining shifts. But the tonnage demands were greater than ever. Consequently the stopes were drawn of their ore too quickly.

As a result of the above conditions a modification of the shrinkage system previously employed was introduced into the last eight shrinkage stopes on the 1350 level.

As a stope is mined upwards, a "dog-hole" drift 5 feet long is driven in the vein and toward the next stope from the manway of the operating stope at a height of 35 or 40 feet above the rail. A jackhammer is used in this operation. Successive dog holes are driven at 35-foot vertical intervals as the stope is advanced to completion (see fig. 11, stopes 28 and 30).

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When the following stope is started a 5-foot pillar of ore is left between the manway of the recently completed stope and the new stope. As shown in 28 stope on Figure 11, the stope is mined inclined until the first dog hole is reached. Then the angle of attack is reversed, and the other end of the stope is advanced and a new dog hole driven. This is repeated, as shown in Figure 11, until the stope is completed. The advantage of this method is that the manway is protected even if the last completed stope has been drawn empty. Disadvantages are:

1. The closing off of the manway from above while advancing upward to hole into the next dog hole. This is especially disadvantageous when the stope is holing into the second and third dog holes because the only entrance is now from below. Ventilation is also restricted.
2. About 4 per cent of the stope to be mined is left as pillars which rarely will be recovered.

The above method has recently been modified by reducing the thickness of the pillar from 5 to 2 feet. A pillar this size is naturally somewhat shattered and will spall off when the two stopes are drawn.

As the length of dog hole is shortened to 2 feet, the short drift need not be driven from the manway side. In stoping upward, when the dog-hole position is reached, the miner now drills with a stoper about six inclined holes from the stope. Upon blasting, this round "bells" the ground away from the manway about 2.5 feet. Thus half of a shift of stoper drilling suffices where formerly two jackhammer rounds were necessary.

Per Cent Extraction

As shrinkage stoping is practiced on the 1350 and 1500 level, no pillars are used except in the modified method employed in certain 1350-level stopes. As the conditions necessitating the modified method no longer exist, the new shrinkage stopes are being opened as shown in Figure 10. Thus 100 per cent of ore in place is extracted. Also the percentage of recovery of the broken ore that has piled up between the chutes is high. There is considerable dilution of the ore by waste spalling from the walls. In addition, as the levels above have been caved, some waste gets mixed with the ore in the final drawing period. All the ore hoisted at present is passed over a picking belt and hand-sorted. From 15 to 18 per cent of the material hoisted is thus rejected.

Transportation, sorting, and loading cost is 20 cents per ton of ore shipped. This cost includes hauling the ore by motor from the ore pocket at the collar of the shaft to the ore bin, sorting, and loading into railroad cars.

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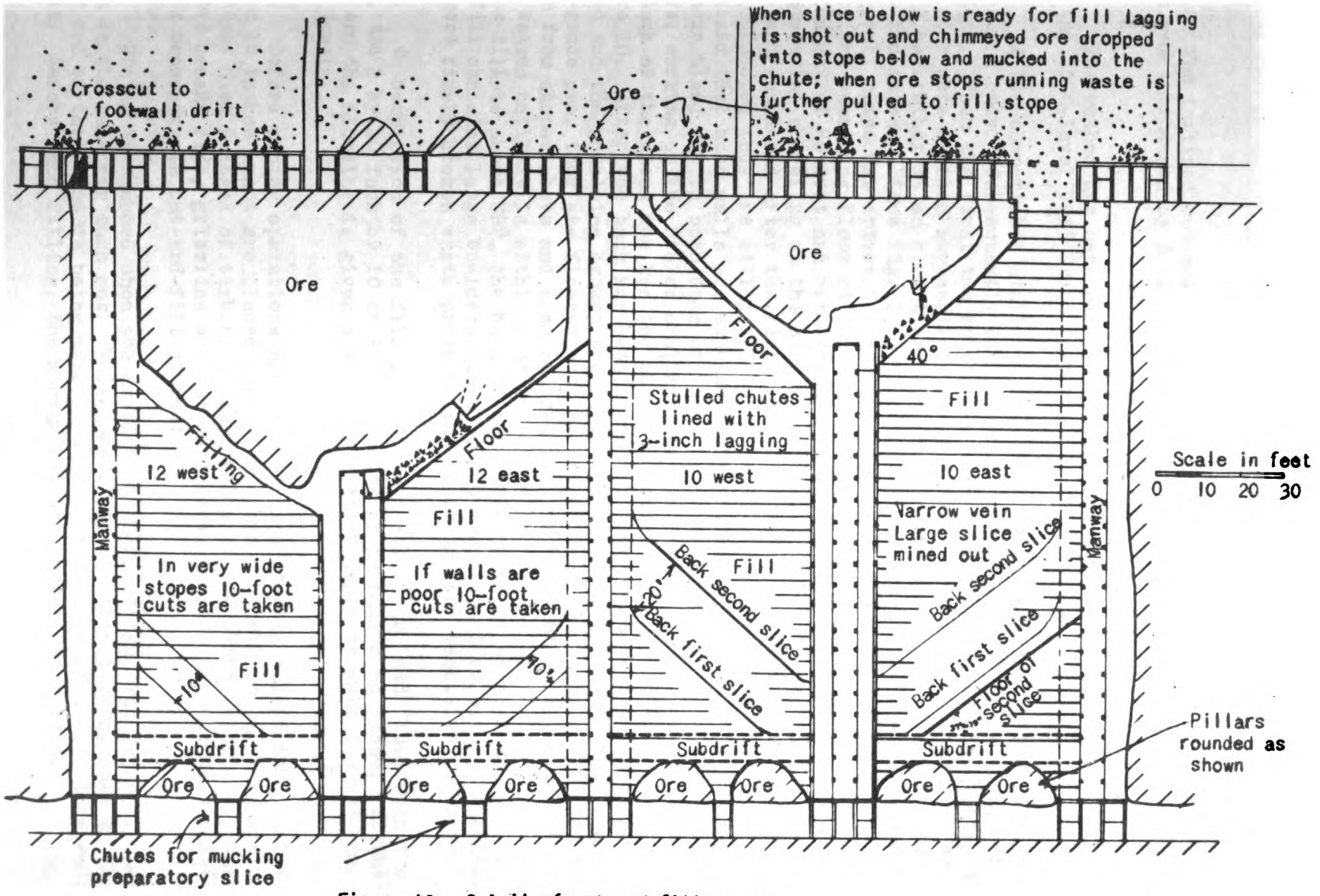

Figure 12.- Detail of cut and fill stopes

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Inclined Cut-and-Fill

The inclined cut-and-fill method as now practiced at the Eighty-Five mines was introduced by A. J. Balmforth, mine superintendent, several years ago.

The method has been employed to date on six stopes, five of which are over the 1500-level West drift (see fig. 2).

A two-compartment stull raise (fig. 6 b) was started at the near limit of the ore body. Subsequently six more raises were started at intervals of 140 feet. When the first two raises were holed through to the 1350 level, the remainder were started successively. Meanwhile, sublevel drifts were driven toward each other at an elevation of 20 feet above the rail to connect the first two raises. (See fig. 12.) Midway between the raises a three-compartment stull raise was driven to an elevation of 26 feet. In the middle of each of the two remaining floor pillars of the stope, untimbered raises were driven to a height of 26 feet and were discontinued. The latter two raises are driven to reduce the shoveling on the sill floor of the stope and are filled with waste and abandoned after the first slice is finished. After the subdrifts have connected they are silled out to the full width of the ore, care being exercised that both walls are exposed. Two successive 5-foot inclined cuts are next mined out in both ends of the stope. If the walls are firm the miners, working on the broken ore, mine one or two additional 5-foot cuts. The back of the stope now has the shape indicated as "end of first slice," Figure 10. After the back has been thoroughly barred down, the chutes are pulled and the remaining broken ore is mucked out. If the sill pillars are high-grade ore, they are floored over before filling the stope. Waste is now drawn from stopes above and also secured from development work through the two fill raises until it is within 2 or 3 feet of the back. As the angle of repose of the fill is about 39°, the back of the stope is mined at that angle. If the fill contains large bowlders, development waste is added before flooring. The more finely broken waste protects the flooring from excessive breakage.

On completion of the fill 2 by 10 inch sills are laid flush with the fill, and 2 by 10 inch by 10 or 12 foot flooring is nailed to these. At the high end the flooring is given a slightly steeper angle to make the ore run more freely.

Before mining operations are started grizzlies of 8 by 8 inch timber spaced 10 inches apart are placed over each chute. Ninety-pound rails are sometimes used instead of timber. Occasionally on the first few slices, when the ore breaks fine, grizzlies are not necessary. Almost no secondary blasting is necessary in cut-and-fill stopes.

Mining is resumed upon the completion of flooring. The cycle of operations is the same for each cut. While one slice is being filled, that on the other end of the stope is being mined. A uniform tonnage of ore is thus maintained. Before each filling, the manway and two chutes are raised.

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When the stope approaches the level above, both slices are mined so that the top of each just reaches the level. In two or three cuts one side of the stope is completed, the direction of retreat being toward the ore chute (see fig. 13). Both overhand mining and brow slabbing are practiced in this operation. The slice is completed when the brow approaches a point directly over the ore chute. The slice is then filled.

As the final slice, when completed, is a rather large one, the hanging wall may slough or the timbers of the old stope overhead may suddenly drop. The floor is therefore given a steep angle (45°) so that the ore will run directly into the chute. Thus mucking is eliminated. The miners are safe on the level above, as most of the pillar is brow-slabbed. In this last slice no grizzly is used over the chute, the ore being pulled only as is necessary until the entire slice is completed, when all the broken ore is drawn. It is after this operation that, as in shrinkage stoping, the broken ore remaining in the stope overhead is partly recovered.

The opposite side of the stope is now mined similarly, the direction of retreat being toward the ore chute. The bottom of the level on this side, however, is stulled with 8 by 8 inch timber and lagged to protect the men below. To date one stope has been completed as described. The walls were firmer than anticipated, and the final pillar left over the ore chute was only 3 feet wide and 14 feet high (deep). The walls stand better than expected because the vein is narrow (5 feet) and because all the mined-out stopes above are completely filled with waste.

During the completion of 10 stope, 300 tons of high-grade ore was recovered from the old stope above. The lagging between the chutes was shot out and the piles of broken ore were recovered.

When the vein is narrow and the walls firm, the height of a cut is increased. Thus shrinkage practice is approximated during part of the mining operation. This method has been referred to as semi-shrinkage.[3]

Hand-rotated stopers are used in overhand slicing, and pluggers in brow-slabbing and plugging bowlders.

The cut-and-fill method is superior to shrinkage stoping in that a cleaner product is obtained. All ore broken in the stopes is recovered. Almost 100 per cent of the ore in floor pillars is recovered.

The disadvantages of the method are the greater amount of development work necessary, and the longer time required to bring the stope into production.

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3 See description in Lavender, H. M., Mining Methods at the Campbell Mine of the Calumet and Arizona Mining Co., Warren, Ariz., Information Circular 6289, Bureau of Mines, 1930, 18 pp.

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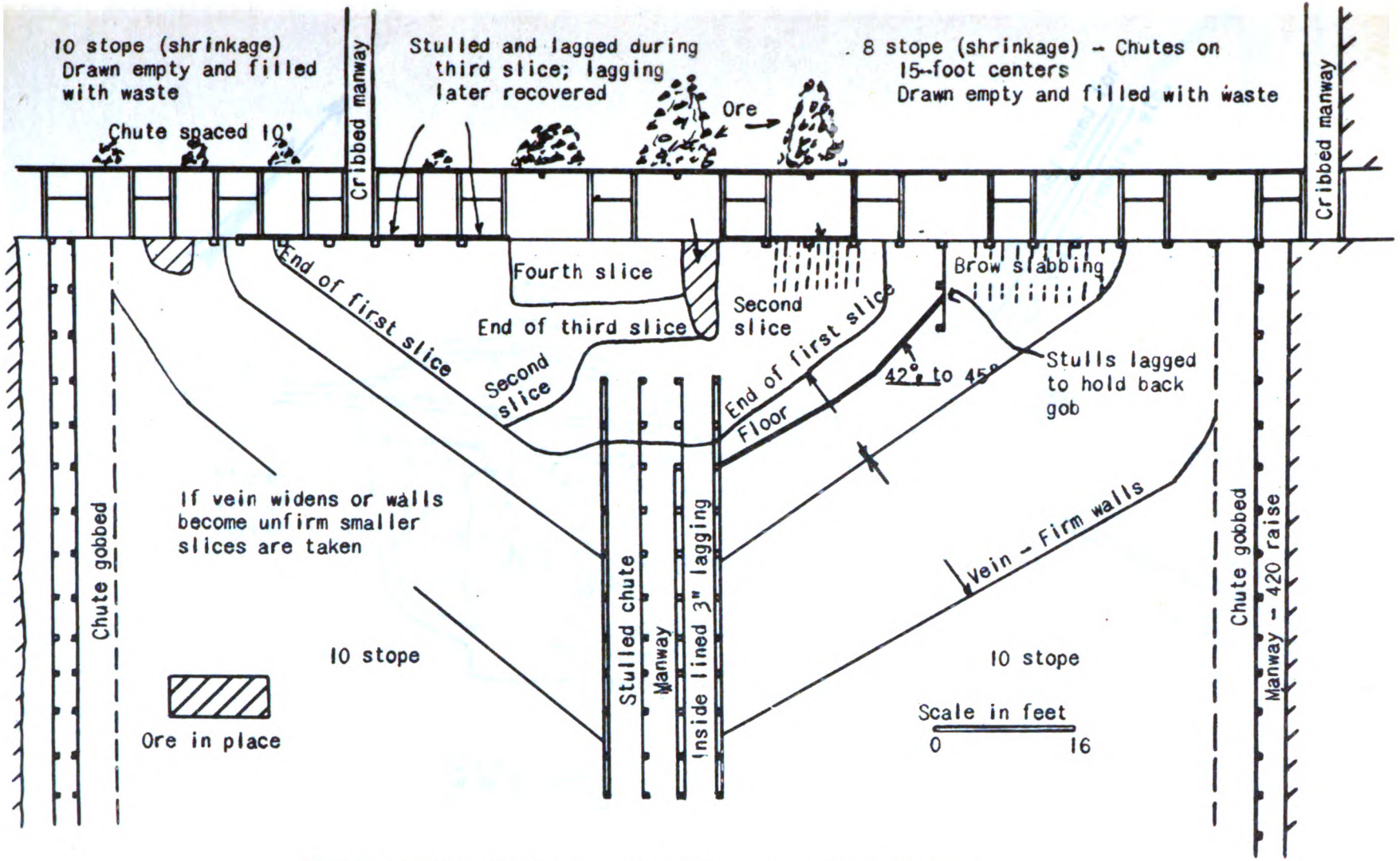

Figure 13.- Longitudinal section of top of cut-and-fill stope

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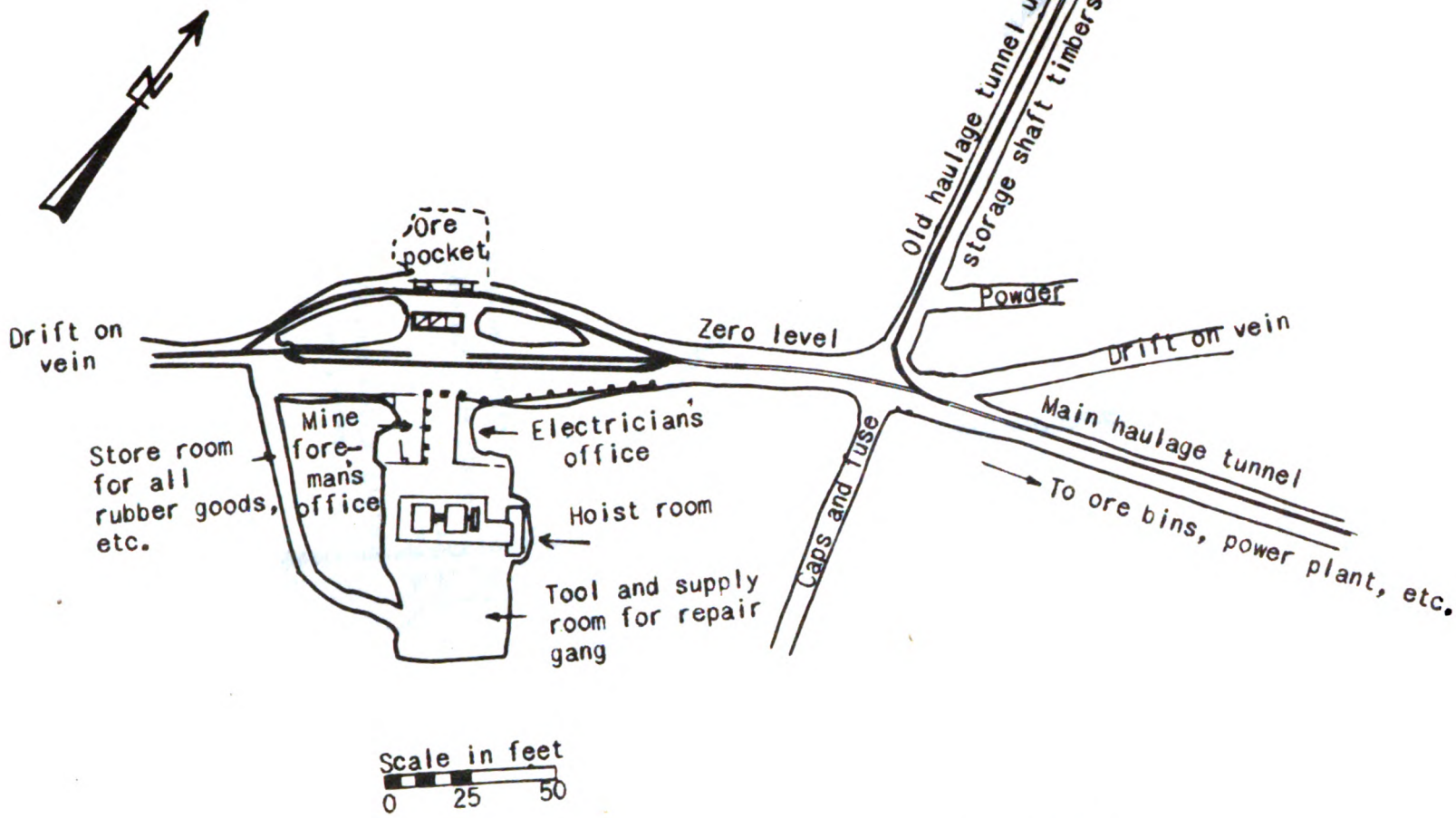

Figure 14.- Plan of zero level at collar of Emerald shaft

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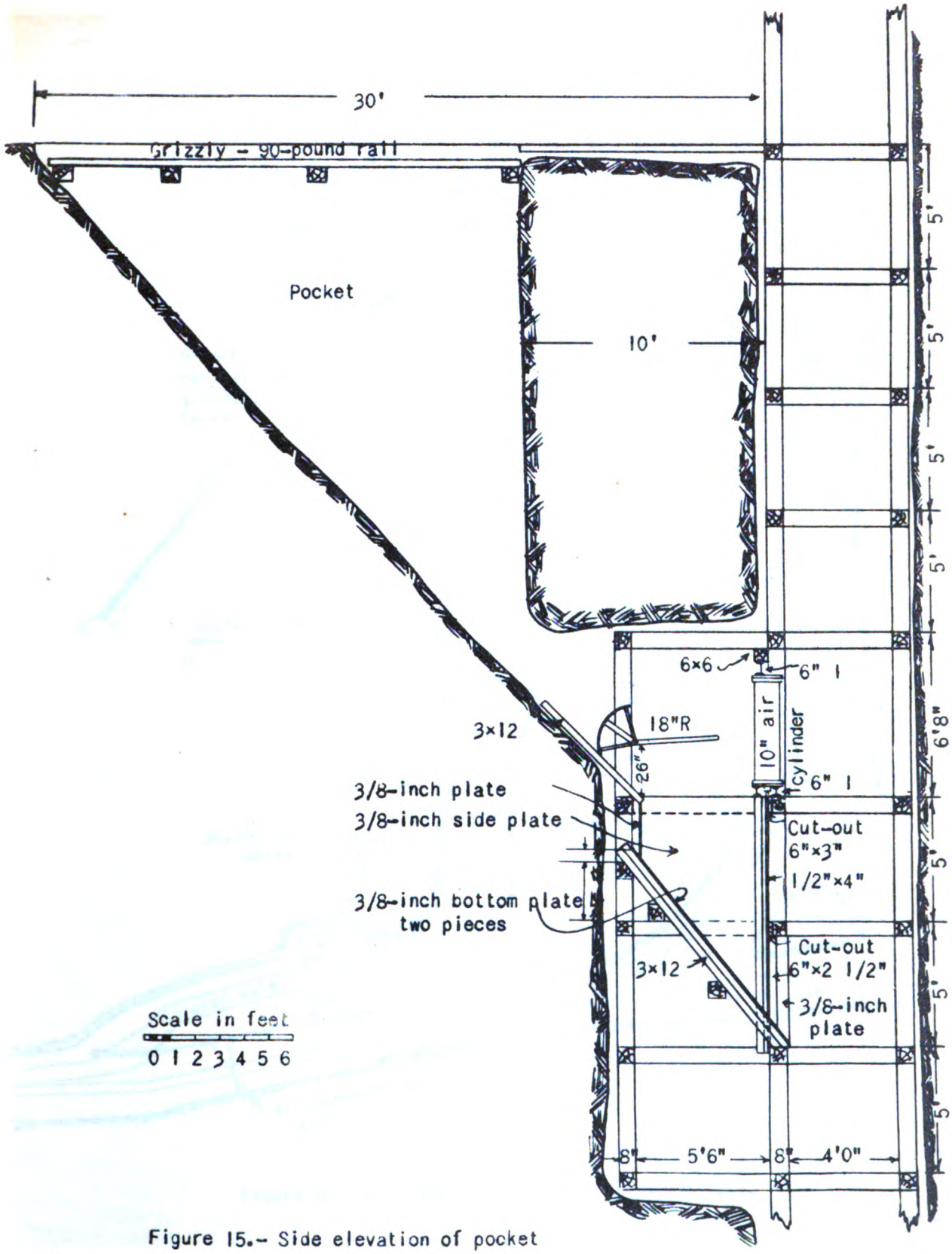

Figure 15.- Side elevation of pocket

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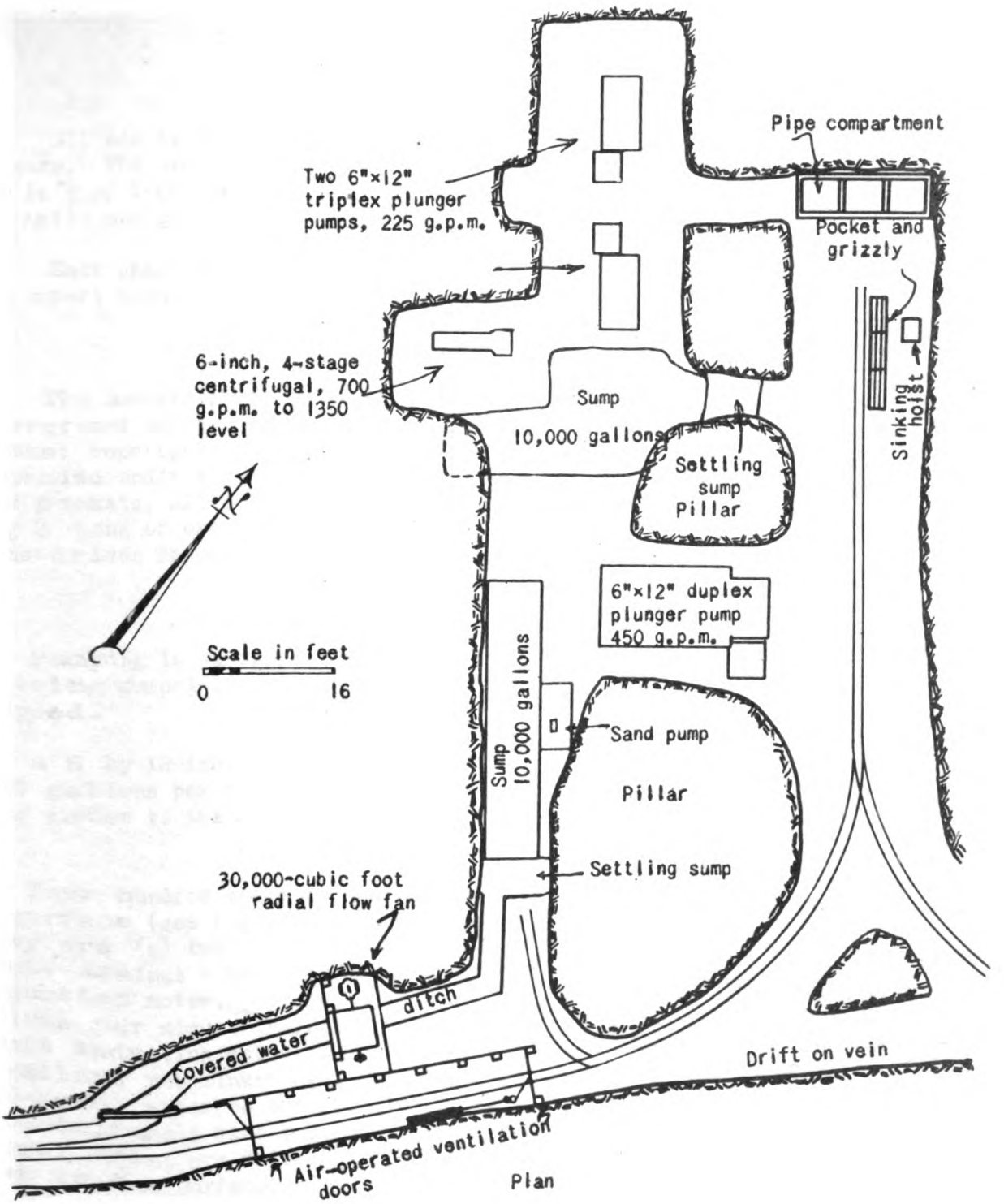

Figure 16.- Pump and fan station of 1650-foot level

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Underground Transportation

All ore is hand-trammed in 16 cubic foot, 1-ton, roller-bearing, end-dump cars. The average tramming distance of ore is 1,200 feet. The track grade is ½ of 1 per cent in favor of the load. Track gage is 18 inches; 16-pound rails are used.

Each shaft station has an ore pocket. Ninety-pound rails spaced 10 inches apart form a grizzly over the pocket.

Hoisting

The Emerald shaft is equipped with a geared electric hoist installed in an underground hoist room on the adit level (fig. 14). The rope load capacity is 8 tons; rope speed is 1,000 feet per minute. The man cage holds 9 men, and is suspended under the skip. Ore is hoisted in balance in 3-ton skips from the various pockets, all of which are equipped with air-controlled skip hoppers holding 3 tons of ore (fig. 15). Ore is handled on the adit level by a 3-ton gasoline-driven locomotive, pulling five 2½ ton V-bottomed side-dump cars.

Pumping

Pumping is an operation of major importance, a total of 450 gallons a minute being pumped to the surface. The cost of this is 46 cents per ton of ore shipped.

A 5 by 12 inch triplex pump, with a capacity of 120 gallons per minute, pumps 50 gallons per minute from the 750 level to the surface. This supplies drinking water to the townsite of Valedon and boiler feed-water to the power plant.

Four hundred and fifty gallons per minute is pumped from the 1650 level to the surface (see fig. 16). Three units can each handle this flow independently. They are (a) two 6 by 12 inch triplex pumps, with a capacity of 450 gallons per minute against a head of 2,200 feet, which are driven by a 200-hp., 440-volt induction motor; (b) one duplex, 6 by 18 inch pump, with a capacity of 440 gallons per minute against a head of 2,200 feet, driven by a 250-hp., 2,200-volt induction motor; (c) one 6-inch centrifugal pump having a capacity of 700 gallons per minute against a head of 700 feet. This centrifugal pumps the 1650-level water to the 1350-level sump. At the 1350 are two tandem, 6-inch centrifugals with a capacity of 700 gllons per minute against a head of 1,400 feet; these are driven by a 250-hp., 440-volt induction motor, and lift the water to the surface.

All water coming from above the 1350 level can be diverted to the sump there and lifted to the surface by one triplex 5 by 12 inch pump having a capacity of 120 gallons per minute against a head of 1,600 feet. This pump is driven by a 100-hp., 440-volt induction motor.

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Two extra pumps in first-class condition are in reserve. One is a deep-well turbine pump, 6-stage, capacity 1,400 gallons per minute against a head of 336 feet; driven by a 200-hp., 440-volt induction motor. The other is a 6-inch centrifugal, capacity 700 gallons per minute against a head of 700 feet; driven by a 250-hp., 440-volt induction motor.

The temperature of the water at the 1650 level face is 104° F. The water is not acid.

For fire protection a 10,000-gallon tank on the hillside back of camp is kept full of water.

Ventilation

Until mining operations reached the 1350 level, a fairly satisfactory natural ventilation system existed. On the east side of the mine the Lower Dundee shaft was strongly downcast; on the west side an air shaft, the Superior shaft, and numerous old openings strongly upcast. Development on the 1500 and 1650 levels was continued using natural ventilation. Although natural ventilation removed the smoke and gas from the mine, the heat and humidity on the bottom levels were oppressive. While developing the 1650 level, West drift, a flow of water with a temperature of 104° F. was encountered. This water, flowing to the 1650 pump station, further increased the already high temperatures. Accordingly a system of mechanical ventilation was started. A 30,000 cubic foot capacity radial-flow fan was installed on the 1650 level (see fig. 16) and suitable doors were built to control the flow of air (figs. 17 and 18). The installation of the fan accelerated the down draft in the Emerald shaft and the Lower Dundee and upcast to the other side of the mine to the new Jim Crow shaft. Temperatures have been reduced and working conditions improved by the increased velocity and volume of air. Conditions at the development faces of the 1500 and 1650 levels are still fairly oppressive; this condition will be remedied when connections are made with the Jim Crow shaft.

Auxiliary air and electrically-driven booster fans serve the development faces off the main air courses. The following chart shows the temperatures and per cent saturation at several places in the mine.

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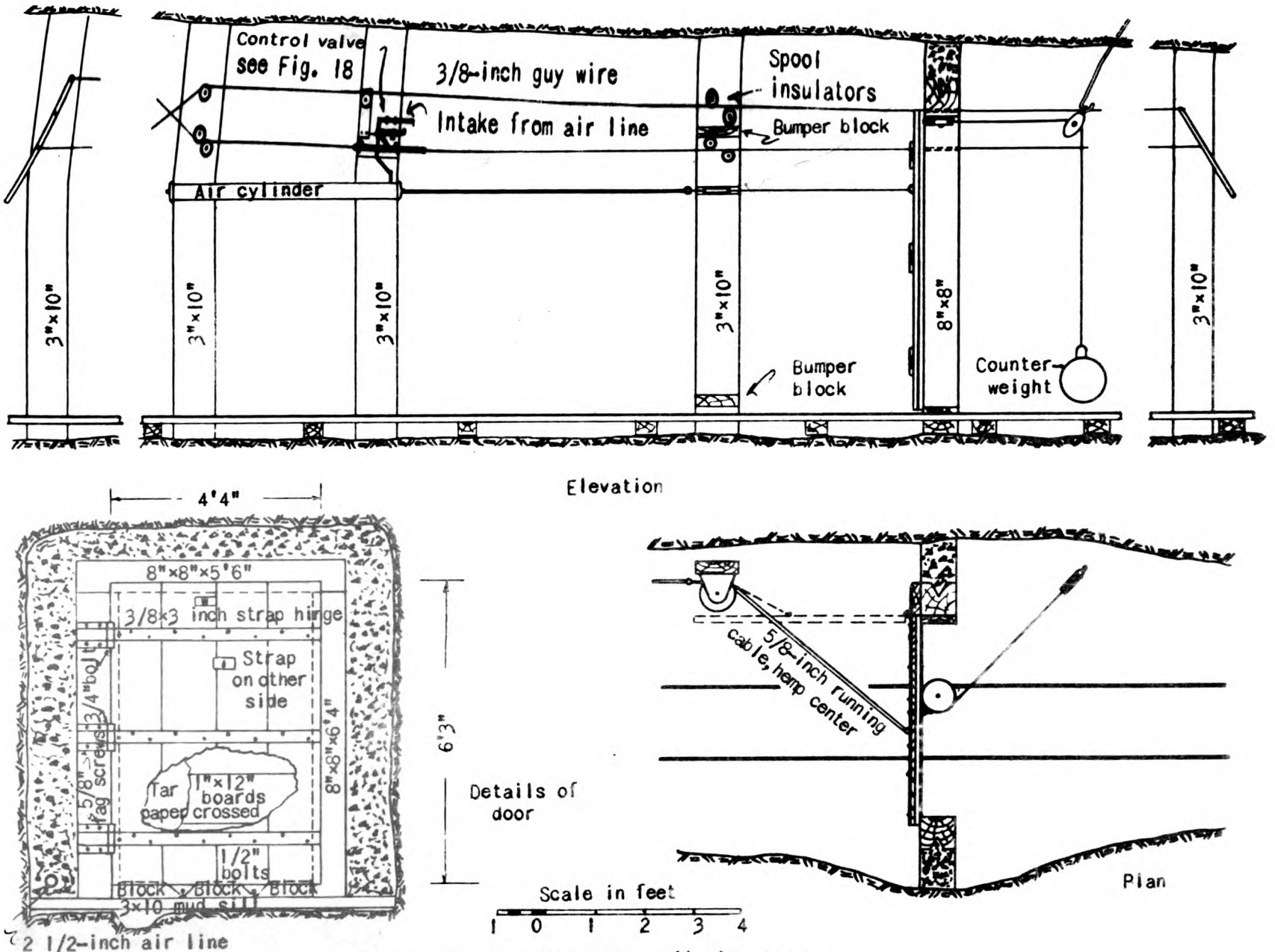

Figure 17.-- Air-operated ventilation doors

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Temperature and humidity at various locations in the Eighty-Five mine

Level	Place	Temperature, °F. Dry	Temperature, °F. Wet	Relative humidity, per cent	Remarks
Zero	Near entrance	79	66	50	9:00 a.m. Sept. 10, weather cloudy and cool.
"	Shaft collar	72	69	86	
1650	Jim Crow shaft station[1]	84	82	92	Compressed air line open, auxiliary blower running.
1350	32 stope	84	84	100	
1500	Face West drift	87½	87½	100	
1650	Pump station	80	80	100	Temperature of water from 1650 West drift, 92° F.
1650	1070 raise	86	86	100	Water temperature 100°F. in main air circuit.
1650	50 feet west of 1070 raise	97	97	100	Just west of main air circuit.
1650	West drift at face	94	94	100	Water temperature, 103.5° F.

1 Not yet connected to main 1650 level and Emeral shaft.

WAGE, CONTRACT, AND BONUS SYSTEM

The following is the wage scale for the principal underground labor classifications:

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Wage scale for the principal underground labor classifications

Classification	Period				
	11-1-1923 to 9-30-1928 also 5-1-1930 to 6-1-1930	10-1-1928 to 1-31-1929	2-1-1929 to 2-28-1929	3-1-1929 to 3-31-1929	4-1-1929 to 5-1-1930
Miner, using Leyner	4.02	4.40	4.60	4.80	5.00
Miner, using stoper	3.30	3.65	3.80	3.95	4.10
Timberman	3.52	3.85	4.05	4.20	4.40
Mucker	2.86	3.15	3.30	3.45	3.60
Trammer	2.86	3.15	3.30	3.45	3.60
Ore sorter	2.64	2.90	3.05	3.15	3.30
Trackman	3.52	3.85	4.05	4.20	4.40
Pipeman	3.52	3.85	4.05	4.20	4.40
Pumpman	4.29	4.70	4.95	5.15	5.35
Hoist engineer--85 shaft.	6.00	6.60	6.90	7.20	7.50
Shaft miner	4.40	4.85	5.05	5.30	5.50
Jigger boss	5.00	- -	- -	6.00	6.25

All labor underground and all unskilled labor on surface is Mexican.

All development work is on a contract rate per foot of advance. Size of opening, class of ground, and working conditions control the variation in price per foot. In drifts and crosscuts the drilling and mucking contracts are separate. Tramming from stopes is on contract. A certain number of cars trammed per day is used as a standard and the trammers are paid 15 cents per car for every car over standard.

Stoping is contracted at a certain rate per yard. The so-called "yard" is a yard long, a yard high, and "vein width" wide. All contractors are guaranteed daily wages.

Shaft sinking is done on contract at a given rate per foot of advance. Only the men at the bottom participate in the bonus.

Standards are set by the chief engineer, subject to approval by the superintendent of the mine. Bonus is paid twice a month on measurements by the engineering department.

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SAFETY WORK

Safety work is under the direction of the safety inspector of the parent company. He makes inspections and recommendations from time to time. The mine superintendent supervises conditions in the absence of the safety inspector.

First-aid instruction is given on the occasion of the visits of the Bureau of Mines instruction car.

First-aid equipment is maintained at the collar of the shaft.

All bosses receive a bonus for accident prevention, based on 1,000 shifts worked.

COSTS

Development costs, per foot

Working place	Labor, including bonus	Explosives	Timber	Air drills	Other supplies	Total	Footage
1465 crosscut, 650 level	$4.29	$1.83	- -	$2.50	- -	$8.62	571.5
West drift, 1650 level..	5.98	2.38	- -	2.75	- -	11.11	1,900.0
1780 raise, 750 level ..	8.60	2.97	$0.82	5.05	$0.11	17.55	100.0

Note: The following costs include direct charges only. The costs for 1465 crosscut, 650 level, are representative. The material cut was hard andesite. The face was 1,000 feet off the main air course, and ventilation was by compressed air.

The 1650-level West drift is in quartz and quartz-seamed monzonite. The face was 900 feet from main air current; two booster fans served for ventilation. A flow of 250 gallons of water per minute at 104° F. made working conditions at the face oppressive. The costs are therefore not representative of the average drift.

The 1780 raise, 750 level was a two-compartment stull raise in quartz, driven from a sublevel.

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Stope costs per ton

Stope number	Level	Type of stope	Year	Bonus	Labor including bonus	Explosives	Timber	Air drills	Other supplies	Total	Tons
10	1500	Cut-and-fill	1928, 1929, 1930	$0.00	$1.26	$0.23	$0.20	$0.60	$0.03	$2.37	7,034
26	1050	Cut-and-fill	1929, 1930	.02	1.62	.17	.27	.38	.02	2.46	4,056
26	1350	Shrinkage, modified method using pillars	1928, 1929	.01	.72	.31	.07	.64	.01	1.75	7,332
3	900	Shrinkage, no pillars used	1927	.05	.49	.24	.10	.39	.01	1.23	4,973

Note: The costs of the 900 level, 3 stope, are low for the following reasons: (1) The stope is located on the main downcast air circuit. (2) The vein width averaged 7 feet; the ore was leached, porous quartz. The above costs for 3 stope are, however, representative of costs in oxidized ore bodies on the upper levels of the east side of the mine.
Tonnage in above shrinkage stopes is wet tons drawn from stopes (before sorting).

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Ore extraction expense

August, 1930

Wet tons mined: 7,220

Accounts	Labor	Explo-sives	Timber	Other supplies	Power	Total	Per wet ton	Per wet ton last month (7,074 tons)
Stoping	$ 6,078.78	$2,050.22	$1,678.50	$ 106.21	$4,288.55	$14,202.26	$1.97	$2.02
Maintenance	424.74	- - -	287.53	1.92	- - -	714.19	.10	.14
Tramming	4,251.82	- - -	- - -	601.15	- - -	4,852.97	.67	.67
Engineering and assaying	628.26	- - -	- - -	173.28	36.15	837.69	.11	.10
Station tending and ore pockets	584.98	- - -	- - -	38.10	- - -	623.08	.09	.08
Underground tracks	939.79	- - -	- - -	79.45	- - -	1,019.24	.14	.16
General underground	2,315.11	- - -	- - -	505.15	36.15	2,856.41	.40	.44
Pumping	609.42	- - -	- - -	449.65	2,693.77	3,752.84	.52	.66
Hoisting	709.43	- - -	- - -	461.81	603.30	1,774.54	.24	.25
Sorting and loading ore.	899.70	- - -	- - -	49.75	36.15	985.60	.14	.14
General surface	275.51	- - -	- - -	24.90	216.78	517.19	.07	.07
Ventilation	115.12	17.25	- - -	150.00	21.34	303.71	.04	.02
Total	$17,832.66	$2,067.47	$1,966.03	$2,641.37	$7,932.19	$32,439.72	$4.49	$4.75
Cost per ton	2.47	0.28	0.27	0.37	1.10	4.49		
Wet tons per shift (day labor 4,989 shifts)							1.45	1.36

Note: The ore extraction expense shown above is fairly representative. The tramming cost includes tramming of ore and waste, chute blasting, and supplies such as mine cars, oil, etc.
Over the past five years the development expense has averaged $1 per ton.
Transportation, treatment, general expense, taxes, and miscellaneous average a total of $2.20 per ton.

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The following tabulation shows power costs for the years 1928 and 1929.

Power costs, 1928 and 1929

	1928		1929	
	Total	Per kw. h.	Total	Per kw. h.
Labor	$ 1,306.67	$0.00390	$14,792.75	$0.00260
Fuel oil and supplies	42,068.53	.01257	65,169.68	.01154
Depreciation	4,800.00	.00144	4,800.00	.00085
6 per cent return on investment	7,200.00	.00215	7,200.00	.00130
Total cost	$67,130.20	$0.02006	$91,962.43	$0.01629

This cost includes labor (3 engineers, 3 firemen, and 1 chief engineer), fuel oil, depreciation at 4 per cent on $120,000, and 6 per cent return on investment.

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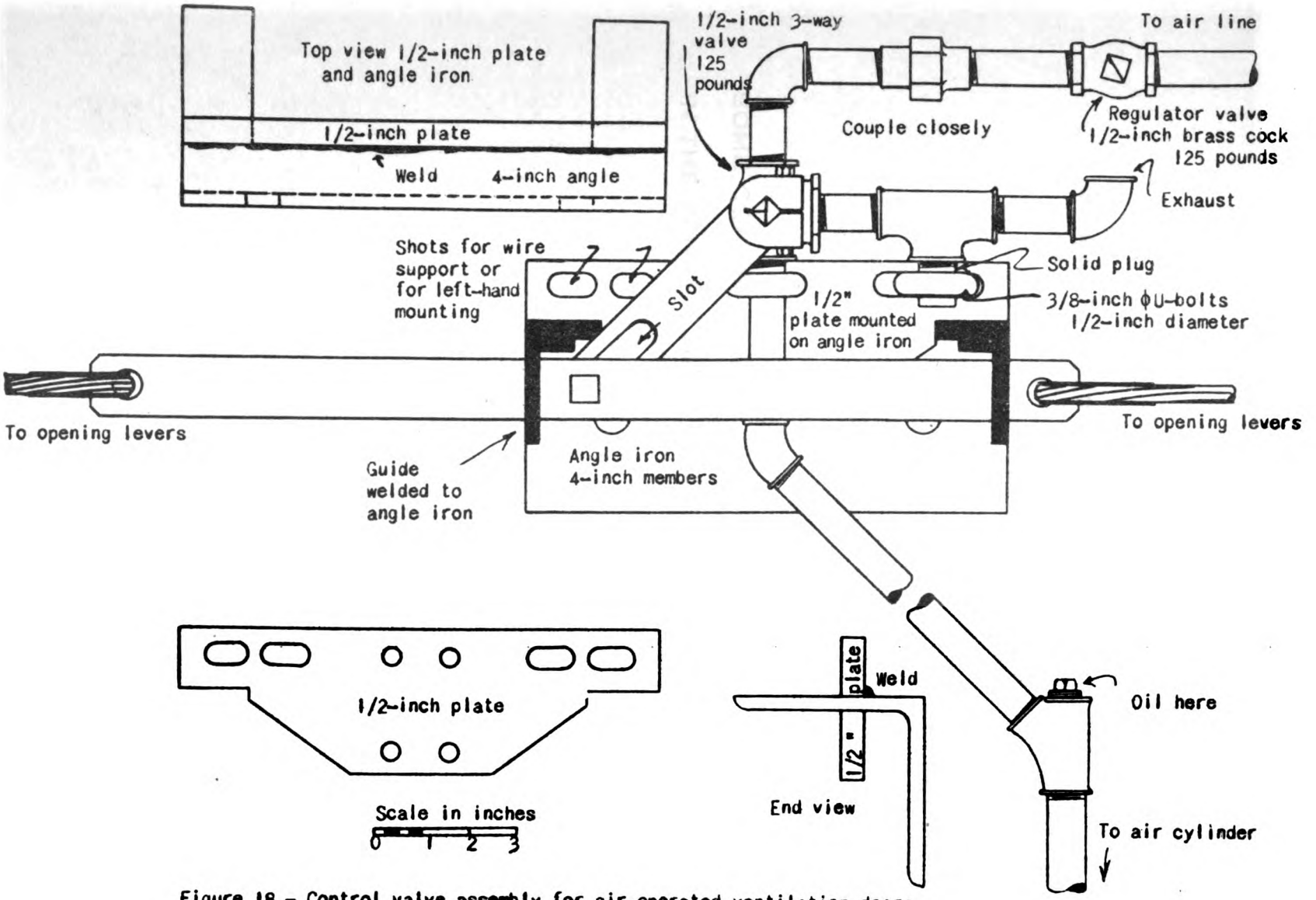

Figure 18.- Control valve assembly for air-operated ventilation doors

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